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TRANSACTIONS

OF THE

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VOL. XLIX.

CONTAINING THE PAPERS AND DISCUSSIONS OF THE SALT LAKE MEETING
AUGUST, 1914.

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PREFACE

This volume, the second of the three volumes which are to contain the papers and discussions contributed to the Institute during the year 1914, comprises papers presented at the Salt Lake meeting of August, 1914, and the discussion thereon. There is also included a discussion of the paper of Theodore Simons, The Evolution of the Round Table for the Treatment of Metalliferous Slimes, which was published in Vol. XLVI of the *Transactions*.

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Proceedings of the One Hundred and Eighth Meeting, Salt Lake City, Utah, August, 1914

The 108th meeting of the Institute was held in Salt Lake City, on Aug. 10 to 14, inclusive, 1914, and appears from past records to be the most largely attended and one of the most successful meetings of the Institute ever held. Three hundred and sixty persons were registered.

To the following Chairmen and members of Local Committees the appreciative thanks of the Institute are due for the admirable arrangements they made for the meeting and for the early publication of these arrangements, which enabled many members to form their plans well in advance.

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J. H. McChrystal J. C. Dick.

SOCIAL AND INSPECTION FEATURES.—General headquarters of the Institute were maintained in two parlors of the Hotel Utah, where an assistant to the Secretary of the Local Committee was in attendance during the entire meeting. Registration facilities were afforded in these headquarters at all times, beginning on the morning of Monday, Aug. 10, and at the same time official badges, booklets, and programs of the meeting were distributed.

At 2:30 p.m., of Aug. 10, there was an organ recital at the Mormon Tabernacle. This was enjoyed by about 300 members and guests of the Institute, and immediately at its conclusion members and guests were taken for a trip around Salt Lake City in sight-seeing motor cars.

On Tuesday, Aug. 11, 269 members and guests left the Union Depot in a complimentary special train and visited the mine of the Utah Copper Co. at Bingham Canyon, where Chairman R. C. Gemmell, of the General Committee, gave a most interesting description of the history, operation, and production of the mine, including operating costs. Luncheon was served on the train *en route* to the Utah Copper Co.'s Magna concentrating plant. After an inspection of this plant, the train conveyed the party to the Garfield smelting plant, and thence to Salt Lake City.

On the same day, the ladies were entertained in the evening at a moving-picture show at the American Theater.

On Wednesday, Aug. 12, a special complimentary train carried 105 members of the Institute to the Murray plant of the American Smelting & Refining Co.; then to the Midvale plant of the U. S. Smelting Co.; and then to the Tooele plant of the International Smelting Co. Luncheon was served on the special train *en route* from Midvale to Tooele.

The ladies were entertained at luncheon at the Salt Lake Country Club and afterward were entertained at tea at the country place of Mrs. Herman A. Prosser.

On Thursday morning, Aug. 13, 112 members went by special complimentary train to Park City, where they visited the Silver King mine and the mill of the Mines Operating Co.

On Thursday afternoon, at 5:45, the members and guests were conveyed by special cars to Saltair Pavilion, where those members who desired to do so had a bath in Great Salt Lake, and afterward dinner was served. The party was carried back to Salt Lake City in special cars.

All day Friday, Aug. 14, was devoted to a trip to Vivian Park in Provo Canyon, where a picnic luncheon was served. On the return trip a short stop was made at the power plant of the Power & Light Co. at Olmstead.

The banquet was served in the Grill Room of the Hotel Utah, and a very enjoyable feature was the opportunity for dancing between courses and after the dinner was over. W. L. Saunders acted as toastmaster, and speeches were made by President Benjamin B. Thayer, Secretary Bradley Stoughton, His Excellency, William Spry, Governor of Utah, and Colonel E. M. Allison.

TECHNICAL SESSIONS.—The first technical session was held on Monday evening, Aug. 10, at 8:30 p. m. An address of welcome was delivered by D. C. Jackling and responded to on behalf of the Institute and its guests by President Benjamin B. Thayer, after which President Thayer introduced R. C. Gemmell as presiding officer for the evening.

The following papers were read by their authors or authors' representatives:

Frederick Laist and Harold Aldrich, Experimental Leaching at Anaconda.

Frederick Laist and F. F. Frick, Precipitation of Copper from Solution at Anaconda.

W. L. Austin, Leaching Copper Products at the Steptoe Works.

W. L. Austin, The Treatment of Copper Ore by Leaching Methods.

Stuart Croasdale, Leaching Experiments on the Ajo Ore.

(These five papers were discussed together by: Lawrence Addicks, R. C. Canby, Stuart Croasdale, D. A. Lyon, and Frederick Laist.)

Frederick Laist and Albert E. Wiggin, The Slime-Concentrating Plant at Anaconda.

Frederick W. Bacorn, An Amendment to Sales's Theory of Ore Deposition.

The following papers were read by title, and discussion was asked for in each case:

Charles W. Goodale, The Drumlummon Mine, Marysville, Mont.

Howland Bancroft, Dip Chart. Discussion, in writing, by Theodore Simons.

Mark N. Alling, Ancient Auriferous Gravel Channels of Sierra County, California.

B. S. Butler and H. D. McCaskey, Copper Ores of the New London Mine.

George A. Packard, Rope Idlers in the Raven Shaft.

James C. Ray then presented an illustrated address entitled, A Geologic and Microscopic Study of the Butte Ores.

The second technical session was held at the Hotel Utah, Tuesday, Aug. 11, at 8:30 p.m., William Wraith presiding. The following papers were presented by their authors or authors' representatives:

W. H. Howard, Electrical Fume Precipitation at Garfield. Discussed (in writing) by E. M. Dunn and F. G. Cottrell.

L. Douglass Anderson, Effects of the Bag House on the Metallurgy of Lead.

Lawrence Addicks, Nodulizing Blast-Furnace Flue Dust. Written discussion by R. M. Draper and James H. Payne.

E. L. Newhouse, Jr., Lead Smelting at East Helena. Discussion by G. C. Riddell and R. C. Canby.

W. W. Norton, A Comparison of the Huntington-Heberlein and Dwight-Lloyd Processes. Discussion by Arthur S. Dwight (in writing) and G. C. Riddell.

G. P. Hulst, The International Lead Refining Plant. Discussion by L. S. Austin and G. P. Hulst.

Irving A. Palmer, Smelting Lead Ores in the Blast Furnace. Discussion by L. D. Anderson (in writing) and L. S. Austin.

Theodore P. Holt, Chloridizing Leaching at Park City. Discussion by F. S. Schmidt and O. C. Ralston.

The following paper was read by title:

H. H. Alexander, The Bag House in Lead Smelting.

The third technical session was held at the Hotel Utah, Wednesday, Aug. 12, at 8:30 p. m., C. W. Whitley presiding. The following papers were presented by their authors or authors' representatives:

Dorsey A. Lyon and Samuel S. Arentz, Losses of Zinc in Mining, Milling, and Smelting.

Richard D. Divine, Separation of Lead, Zinc, and Antimony Oxides.

H. A. Wentworth, Electrostatic Separation at Midvale (read by title only).

S. E. Bretherton, The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process.

(The four preceding papers were then discussed together by James M. Hyde, George W. Riter, S. A. Ionides, S. S. Arentz, O. C. Ralston, D. A. Lyon, Mr. Swart, S. A. Kent, Lawrence Addicks, S. E. Bretherton, and G. B. Wilson.)

Earl S. Bardwell, The Annealing of Cold-Rolled Copper. Discussion by Lawrence Addicks.

E. Horton Jones, Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd.

Dorsey A. Lyon and Robert M. Keeney, Melting of Cathode Copper in the Electric Furnace. Discussion by Lawrence Addicks.

L. O. Howard, Basic-Lined Converter Practice at the Old Dominion Plant. Discussion by E. P. Mathewson.

The following papers were read by title only and discussion was asked for:

C. D. Demond, Economy and Efficiency in Reverberatory Smelting.

C. R. Kuzell and G. H. Wigton, Curves for the Sensible-Heat Capacity of Furnace Gases.

Oscar M. Kuchs, Lead-Matte Converting at Tooele.

The fourth technical session was held at the Hotel Utah, Thursday, Aug. 13, at 9:30 a.m., Sidney J. Jennings presiding.

The following papers were presented in abstract by their authors or authors' representatives:

Ira B. Joralemon, The Ajo Copper-Mining District.

John Van N. Dorr, The Dorr Hydrometallurgical Apparatus.

G. H. Clevenger, Discussion of James Johnston's paper, Mill and Metallurgical Practice of Nipissing Mining Co., Ltd., Cobalt, Ont., Canada. Discussion (in writing) by Thomas Crowe and Allan J. Clark.

Robert H. Bedford and William Hague, Tests of Rock Drills at North Star Mine. Discussion by W. L. Saunders.

E. D. Gardner, Mining Claims Within the National Forests. Discussion by T. C. Hoyt, Asst. Chief Forester, U. S. Forestry Service; H. V. Winchell and D. W. Brunton.

The following papers were read by title only:

Frank R. Van Horn, The Occurrence of Bournonite, Jamesonite, and Calamine at Park City, Utah.

A. W. Allen, The Descriptive Technology of Gold and Silver Metallurgy.

William F. Ward, "Playa" Panning on the Cauca River.

Joseph P. Hodgson, Mining Methods at the Copper Queen Mines.

Robert Livermore, Draining Kerr Lake.

Sidney L. Wise and Walter Strache, The Design, Construction, and Cost of Two Mine Bulkheads.

Carl A. Allen, Methods and Economies in Mining.

R. W. Raymond, Biographical Notice of Louis Janin.

Arthur Crowfoot, Development of the Round Table at Great Falls.

PAPERS

Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd.

BY E. HORTON JONES*, CLIFTON, ARIZ.

(Salt Lake Meeting, August, 1914)

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INTRODUCTION

WE have endeavored in the following "sheets" to give the unit construction costs derived from the building of the Arizona Copper Co.'s new smelter, Clifton, Ariz., starting in February, 1912, and completing February, 1914.[1] In Chapter I—Unit Costs—are to be found the most elementary total unit costs which the accounts provide for. They are usable in the Clifton district. Here too are found the percentages to be added to an estimate for Engineering and General Expense. In Chapter II—Comparative Costs—these elementary costs have been classified averaged and reported as labor and material unit costs. In such form the labor unit costs, when properly applied to similar conditions as these under which they were derived, are usable anywhere. The material unit costs are better disregarded for more accurate estimates and replaced by a newly priced bill of material. In Chapter III Composite Costs are given. They are unit costs built up from several elementary units, and likewise units of larger dimensions and simpler application, valuable for

* Chief Engineer, Arizona Copper Co., Ltd.

[1] The smelter went into service producing copper October, 1913.

checking estimates and obtaining quick approximations of total costs. In Chapters IV, V, and VI are given the Wage Scale, Material Prices, and a description of the conditions surrounding the making of every elementary unit cost, which will enable an estimator to judge of their use under any circumstance. It should be borne in mind that Chapter VI is not a pure description of the plant, nor of any of its parts, such as might be expected in a technical journal, but it is barely enough information for a reader to judge as to the applicability of a unit cost in another place.

These unit costs, as any examination will prove, are not ideal, but actual. They were made by a copper company organization extending over a period of two years. They represent delays in material shipments (serious delays in steel and brick deliveries), delayed plans, changes in plans, labor troubles, with changes in hours and rates, variable weather conditions, and the like. They do represent, however, every cent spent.

Each unit cost has been judged, as it stands, solely on its merits for use by an estimator. Where a unit cost is not given it was of no use, and where given and described, it must be employed accordingly.

During the first few months of construction work only the prime account numbers showing in the sheet were in use. Later, to obtain more accurate figures to measure the work by as it progressed, the decimal account numbers were added. But the decimal accounts were on record only in the engineering costing department at the new smelter, two miles from Clifton. The General Office, at Clifton, neglected all decimal accounts and charged them under the prime numbers. The labor segregation was in all instances made at the new smelter, and every charge for material, from whatever source, passed through the warehouse at the new smelter. The only other source of cost sheet entries was the General Office cash books. Once a month these entries were itemized and sent out in the monthly Cost Sheet issued by the General Office.

CHAPTER I

UNIT COSTS

The costs thus accrued from three sources. The individual labor card was not used until the excavating was well under way and the foundation work had started. Each man, who could, made out his card and told thereon exactly at what and how long he had been working. The labor bosses made out the cards for the Mexicans. The account number was later placed on the card by the time keeper and checked by a competent man in the engineering office. For every bit of material used on the job a requisition was passed through the warehouse, whether steel building or keg of nails. To this a charge number was attached and the requisition was finally checked by a competent man in the Engineering Department.

In regard to cash-book entries, there was little to be done, as all local bills were OK'd and account numbers attached by the Engineering Department in the first instance. The units were compiled by the Engineering Department daily as the work progressed and checked suitably.

Yet owing to the fact that the General Office kept the accounts segregated only under the prime account numbers, the slow development of the idea of making a final accurate unit cost sheet, the delay in using individual time cards and many clerical mistakes, it was finally thought best to check over every labor card and warehouse requisition from the beginning of the job. This was done and the unit costs as they now stand are believed to be accurate.

The making of this cost sheet is largely due to the untiring interest and insistent demand for accuracy upon the part of three members of the smelter construction force, G. H. Ruggles, M. Am. Inst. M. E.; Roy B. Earling, M. Am. Inst. M. E.; and H. F. Adams. Credit is due also to the ever willing assistance given by the Arizona Copper Co.'s general office, under the direction of J. G. Cooper, Cashier.

General Expense

Number	Name of Account	Total	
7001	General expense at Clifton	See page 15	
7004	Personal injuries	" " 15	

Engineering

Engineering and Superintendence at Douglas

7101	Salaries	$39,706.22	
7103	Telegraphing and telephoning	167.55	
7104	Traveling expense	1,984.85	
7105	Miscellaneous expense	1,018.06	
			$42,876.68

Engineering and Superintendence at Clifton

7201	Salaries	$40,587.54	
7202	Furniture and fixtures	149.43	
7203	Telegraphing and telephoning	258.64	
7204	Traveling expense	934.24	
7205	Miscellaneous supplies	2,205.77	
7206	Miscellaneous labor	869.02	
			45,004.64
9000.1	Power plant engineering	$12,768.56	12,768.56
7100	Total engineering expense		$100,649.88
	Total unit cost		5.40 per cent.

This percentage is obtained by dividing the engineering expense by the total cost of the smelter, minus engineering and indirect expense.

$$\frac{100,649.88}{1,864,092.47} = 5.40 \text{ per cent.}$$

7300.—Yard Tracks and Industrial System

Tracks

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
7301	Excavation	$31,311.14	$4,254.86	$35,566.00	55,405 cu. yd.	$0.64
7302	Ties	425.13	10,777.79	11,202.92	10,262 ties	1.09
7303	Rails and rail fastenings	392.00	9,839.79	10,231.79	14,813 ft.	0.69
7304	Frogs and switches	206.50	2,676.95	2,883.45	18 switches	160.19
7305	Laying, surfacing and ballasting	5,873.53	248.88	6,122.41	17,150 ft.	0.36

Trolley System

7306	Poles and setting	1,427.27	1,804.73	3,232.00	106 poles	30.49
.1	Brackets and wiring	1,167.19	1,379.73	2,546.92	7,824 ft.	0.33
.2	Rail bonds	304.31	347.13	651.44	521 bonds	1.25
.3	Lighting	229.66	222.39	452.05	57 lamps	7.93

Rolling Stock

7307	Cars, elec. locomotives, etc	801.29	34,017.63	34,818.92	.....	.....
.1	Calcine car alteration	381.02	4.36	385.38	.....	.....

Trestle Approach to Reverberatory Building

7308	Excavation	359.95	.....	359.95	277 cu. yd.	1.30
.1	Foundations	738.18	1,152.52	1,890.60	254.9 cu. yd.	7.42
.2	Steel structure	.....	.....	13,460.84	163.97 tons	82.09
.3	Woodwork	703.93	768.92	1,472.85	27.65 m.b.m.	53.27

150-Ton Track Scales in Receiving Yard

7309	Excavation	348.91	.....	348.91	388 cu. yd.	0.90
.1	Foundation	545.62	1,146.87	1,692.49	186 "	9.10
.11	Cost and erection	365.95	3,450.86	3,816.81	150 tons	25.45
.30	Scale house	65.66	63.87	129.53	879 cu. ft.	0.15

Bridges, Culverts and Walls

7310	Bridge No. 1 foundation	1,028.26	1,968.38	2,996.64	339.8 cu. yd.	8.82
.1	" steel work	.....	.....	377.40	3.70 tons	102.00
7311	Culvert No. 1 masonry	1,384.85	695.80	2,080.65	354 lin. ft.	5.88
7312	Retaining walls excavation	77.66	.....	77.66	60 cu. yd.	1.29
.1	" " concrete	512.34	734.25	1,246.59	203.5 "	6.13
.2	" " masonry	88.08	47.51	135.59	21.9 "	6.19

40-Ton Track Scales on Calcine Track

7313	Excavation	108.44	0.51	108.95	118 cu. yd.	0.92
.1	Foundation	207.55	193.60	401.15	41.6 "	9.64
.2	Cost and erection	82.99	710.85	793.84	40 tons	19.85
.3	Scale house	100.38	88.21	188.59	879 cu. ft.	0.22

Trestles to Receiving Bins

7314	Excavation	548.18	.....	548.18	589 cu. yd.	0.93
.1	Foundation	2,408.16	3,017.95	5,426.11	754.3 "	7.19
.2	Steel structure	.....	.....	9,269.48	109.35 tons	84.77
.3	Woodwork	572.23	838.11	1,410.34	27.21 m.b.m.	51.83

Total cost—Yard tracks and industrial system..... $156,326.43

7400.—Receiving Bins

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
7401	Excavation	$2,303.11	$39.16	$2,342.27	1,428 cu. yd.	$1.64
7402	Foundation	1,235.51	2,247.70	3,483.21	612.3 "	5 69
7403	Steel structure	.....	.....	29,276.63	353.09 tons	82 92
7404	Gates	901.15	1,984.93	2,886.08	30 gates	96.20
7405	Conveyor No. 1	310.92	2,947.19	3,258.11	97.3 ft.	33.49
.01	Conveyor No. 2	355.19	2,498.03	2,853.22	117.3 ft.	24.33
7407	Lighting	60.87	24.67	85.54	22 drops	3.89
	Total cost—Receiving bins			$44,185.06		

7700.—Crushing Plant

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
7701	Excavation	$689.67	.....	$689.67	609 cu. yd.	$1.13
7702	Foundation	893.15	$1,568.47	2,461.62	220.5 "	11.16
7703	Steel structure	.....	.....	2,420.36	25.07 tons	96.54
.1	Doors, windows and frames	84.00	170.71	254.71	529 sq. ft. opening	0.48
.2	Painting woodwork	15.00	12.33	27.33	70 sq. yd.	0.39
7704	Crushing machinery	392.86	1,093.61	1,486.47	500 cwt.	2.97
.1	Chutes	325.87	338.09	663.96	118.1 cwt.	5.62
7705	Shafting, pulleys and belting	4.17	483.35	487.52	12 lin. ft.	.....
7706	Motor	102.81	513.64	616.45	50 h.p.	12.33
7707	Power wiring	26.55	19.35	45.90	.....	.....
.1	Lighting	76.41	38.22	114.63	8 drops	14.33
	Total cost—Crushing plant			$9,268.62		

7800.—Sampling Plant

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
7801	Excavation	$274.09	$20.97	$295.06	332 cu. yd.	$0.89
7802	Foundation	605.91	649.44	1,255.35	120.7 "	10.40
.1	Concrete ground floor	105.68	263.92	369.60	1,222 sq. ft.	0.30
.2	Reinforced concrete floors	1,050.61	1,080.30	2,130.91	4,244 "	0.50
7803	Steel structure	.....	.....	10,408.12	110.85 tons	93.89
.1	Doors, windows and frames	332.94	564.90	897.84	2,086 sq. ft. opening	0.43
.11	Painting doors and windows	118.94	28.96	147.90	129 sash	1.15
7804	Shafting, pulleys and belting	64.01	1,871.07	1,935.08	85 ft. of shafting	22.76
7805	Meters	186.76	887.57	1,074.33	90 h.p.	11.94
7806	Power wiring	78.11	120.55	198.66	.....	.....
.1	Lighting	184.45	140.57	325.02	36 drops	9.03
7807	Rolls and samplers, cost and erection	1,215.83	7,899.14	9,114.97	1,251.1 cwt.	7.29
.1	Cast iron liners and drying pan, and erection of chutes	1,240.69	1,001.86	2,242.55	270.6 cwt.	8.29
.20	Elevator	20.39	458.22	478.61	1 elevator	478.61
.50	Steel chutes (made at new smelter)	1,995.70	596.65	2,592.35	198 cwt.	13.09
7809	Keystone plate partitions	131.04	338.32	469.36	1,523 sq. ft.	0.31
7810	Alterations of chutes and machinery	128.69	44.34	173.03	.....	.....
	Total cost—Sampling plant			$34,108.74		

7900.—Bedding Plant and Bunker Bins

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
7901	Excavation	$11,577.53	$681.90	$12,259.43	12,319 cu. yd.	$0.99
7902	Foundation	6,256.34	14,513.21	20,769.55	2,809.7 "	7.39
7903	Steel structure	.....	.....	47,404.86	548.71 tons	86.39
7904	Conveyors 7^1–7^3	564.34	6,373.67	6,938.01	380.2 ft.	18.24
.1	" 8^1–8^2–8^3	1,211.01	8,718.98	9,929.99	562 ft.	17.67
.2	" 9^1–9^2–9^3–10^1–10^3	1,912.20	9,756.19	11,668.39	905.5 ft.	12.88
.3	Bunker bin gates	161.58	1,021.64	1,183.22	42 gates	28.17
.4	Chutes for conveyors 7^1 to 10^3 inc	658.91	842.60	1,501.51	276.2 cwt.	5.44
7905	Two reclaimers	3,103.28	30,579.45	33,682.73	2 reclaimers	16,841.37
.1	" wiring	377.21	131.19	508.40	2 "	254.20
7906	Lighting	306.85	127.38	434.23	63 drops	6.89
7907	Transfer car	739.12	3,823.19	4,562.31	1 car	4,562.31
7908	Signal system	67.68	28.74	96.42	.....	.....
	Total cost—Bedding plant and bunker bins			$150,939.05		

8100.—Roasting Plant

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8101	Excavation	$1,547.07	.....	$1,547.07	1,216 cu. yd.	$1.27
8102	Foundation	765.42	$1,069.63	1,835.05	250.5 "	7.33
8103	Steel structure	.....	.....	37,252.67	445.28 tons	83.66
.1	Elevator	.....	.....	2,189.62	5 tons capacity	437.92
8104	Roasters, cost and erection	3,716.94	56,326.09	60,043.03	8 roasters	7,505.38
.1	" alterations	620.49	81.69	702.18	.....	.....
8105	" brickwork	4,730.85	12,336.35	17,067.20	16,104 cu. ft.	1.06
.01	" unloading brick	363.14	.....	363.14	1,231.01 tons	0.29
.02	" centering for brickwork	1,389.62	588.25	1,977.87	8 roasters	247.23
8106.01	Roaster flue, spouts	116.52	568.99	685.51	10 spouts	68.55
.02	" " tile work	374.82	400.36	775.18	2,365 cu. ft.	0.33
.03	" " painting inside	8.50	3.97	12.47	73 sq. yd.	0.17
8107	Shafting, pulleys and belting	118.24	1,999.89	2,118.13	164 lin. ft.	12.92
8108	Motor	277.08	463.96	741.04	30 h.p.	24.70
8109	Lighting	340.64	157.70	498.34	67 drops	7.44
8112	Motor-driven fans	77.69	1,405.91	1,483.60	2 fans	741.80
.1	Blast pipe from fans to roaster	1,569.62	656.62	2,226.24	240 ft.	9.28
8113	Conveyor No. 12	164.15	805.05	969.20	51.2 ft.	18.93
.1	" 13^1 and 13^2	476.29	3,472.33	3,948.62	217.3 ft.	18.17
.2	Stile over conveyors	98.91	199.80	298.71	2.40 tons	124.46
	Total cost—Roasting plant			$136,734.87		

8120.—Roaster Dust Chamber

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8121	Excavation	$918.74	$194.09	$1,112.83	1,194 cu. yd.	$0.93
8122	Foundation	1,049.71	1,775.05	2,824.76	472.9 "	5.97
8123	Steel structure	.....	.....	34,745.41	415.68 tons	83.59
.01	Wire baffles	523.63	4,758.23	5,281.86	604.8 c. wires	8.73
.1	Tile work	2,268.29	2,585.45	4,853.74	14,980 cu. ft.	0.33
.11	Unloading tile	307.72	.....	307.72	525.05 tons	0.59
.2	Painting outside	304.75	115.82	420.57	2,851 sq. yd.	0.15
.3	Painting inside	93.50	24.37	117.87	950 "	0.12
	Total cost—Roaster dust chamber			$49,664.76		

8300.—Reverberatory Plant

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8301	Excavation	$1,417.88	$59.78	$1,477.66	1,890 cu. yd.	$0.78
.01	Back filling	2,742.44	13.11	2,755.55	3,679 "	0.75
8302	Foundation	7,715.96	15,044.09	22,760.05	2,810 "	8.10
.1	Concrete counterweights.	88.22	79.27	167.49	8.5 "	19.71
8303	Steel structure	.....	.....	40,799.76	461.09 tons	88.48
8304	Reverberatories, brickwork	8,402.75	34,963.88	43,366.63	29,680 cu. ft.	1.46
.01	" unloading brick	934.78	329.71	1,264.49	2,279.49 tons	0.55
.02	" centering	859.77	547.78	1,407.55	3 reverberatories	469.18
.05	" rehandling brick	1,503.85	.....	1,503.85	1,530.92 tons	0.98
.1	" steel work	1,366.38	14,552.71	15,919.09	240 tons	66.33
.2	" silica fill	3,715.66	5,689.41	9,405.07	1,231.49 tons	7.64
.3	" hoppers and chutes.	150.08	1,263.59	1,413.67	403.35 cwt.	3.51
8305	Cross and header flues, brickwork	3,296.65	15,098.97	18,395.62	18,500 cu. ft.	0.99
.01	" unloading brick	403.77	93.83	497.60	1,329.23 tons	0.37
.02	" centering	316.84	125.97	442.81	3 sets	147.60
.2	" painting brick	46.88	21.45	68.33	473 sq. yd.	0.15
8306	Flues from boilers to reverberatory, flue excavation	19.86	.....	19.86	15 cu. yd.	1.32
.1	" foundation	74.22	74.02	148.24	9.2 "	16.11
.2	" steel structure	.....	.....	2,815.32	34.78 tons	80.95
8307	Reverb. boiler bldg. exca'n	283.14	.....	283.14	306 cu. yd.	0.93
.01	Waste heat boilers "	213.44	.....	213.44	129 "	1.65
.02	Oil-fired boilers "	73.60	.....	73.60	97 "	0.76
.03	Boiler feed pumps "	591.28	10.16	601.44	659 "	0.91
.04	Backfill, back of boiler wall	518.10	30.00	548.10	972 "	0.56
.1	Boiler building foundations	2,181.08	3,846.18	6,027.26	573.7 "	10.51
.11	Waste heat boilers "	789.52	793.97	1,583.49	138.7 "	11.42
.12	Oil-fired boilers "	350.45	686.29	1,036.74	100. "	10.37
.13	Feed pumps foundation	678.73	1,428.97	2,107.70	214.7 "	9.82
.2	Floor over slag track cut	1,018.17	1,687.29	2,705.46	7,676 sq ft.	0.35
.3	Floor around boilers	276.29	664.82	941.11	2,705 "	0.35
8308	Boiler building, steel structure	.....	.....	25,839.85	292.03 tons	88.48
.5	Platform and brackets	1,317.18	2,556.45	3,873.63	29.50 tons	131.30
8309	Waste heat boilers, inc. all steel	4,078.19	39,335.30	42,413.49	7 boilers	6,059.07
.01	" " brickwork	3,878.57	13,749.97	17,628.54	18,025 cu. ft.	0.98
.02	" " unloading brick	510.74	8.17	518.91	1,073.74 tons	0.48
.03	" " painting	140.53	84.29	224.82	1,190 sq. yd.	0.19
.05	" " rehandling brick	185.76	.....	185.76	1,135.57 tons	0.16
.1	Oil-fired boilers, inc. all steel	1,892.22	13,125.78	15,018.00	3 boilers	5,006.00
.11	" " brickwork	1,697.31	1,154.62	2,851.93	6,561 cu. ft.	0.43
.12	" " unloading brick	168.44	15.50	183.94	228.11 tons	0.81
.13	" " painting	124.59	24.16	148.75	438 sq. yd.	0.34
.15	" " rehandling brick	137.06	.....	137.06	413.35 tons	0.33
8310	Superheate rswaste heat boilers	654.41	8,288.71	8,943.12	7 heaters	1,277.58
.1	" oil-fired boilers	280.28	2,756.41	3,036.69	3 "	1,012.23
8312	Misc. piping boilers and reverb. bldg.	524.15	1,409.85	1,934.00	.....	.....
.1	Feed piping from heating plant to feed pumps exca'n	1,039.91	51.51	1,091.42	1,296 cu. yd.	0.84
.11	" " conduit and pipe	386.25	2,757.79	3,144.04	557 ft.	5.6

8300.—Reverberatory Plant (*Continued*)

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8312.20	Feed piping from pumps to boilers	1,060.53	3,041.00	4,101.53	1,093 ft.	3.75
.50	Blow-off piping and drum.	145.32	527.77	673.09	.....	.....
8313	Wiring electrical feed pumps	135.45	177.89	313.34	2 pumps	156.67
.1	Lighting for reverb. and boiler bldg	612.30	473.19	1,085.49	104 drops	10.44
8314	Slag launders	433.20	1,157.80	1,591.00	72 ft.	22.10
8315	Matte launders	142.66	1,209.08	1,351.74	304 ft.	4.45
8316	Six No. 14 Wilgus oil systems	111.48	1,862.29	1,973.77	6 pumps	328.96
8317	Two electrical feed pumps.	240.25	5,633.97	5,874.22	2 "	2,937.11
.1	One steam feed pump	37.27	499.24	536.51	1 "	536.51
.2	Crawls and chain block in feed pump house	2.00	112.05	114.05	.....	.....
8318	Fettling system	1,463.13	1,943.08	3,406.21	3 furnaces	1,135.40
	Total cost—Reverberatory plant			$328,945.02		

8400.—Converter Plant

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8401	Excavation	5,163.74	567.37	5,731.11	6,330 cu. yd.	$0.91
8402	Foundation	1,796.56	2,860.16	4,656.72	776.9 cu. yd.	6.00
8403	Steel structure	.....	.....	69,359.60	783.86 tons	88.48
8404	Converter stands—Excavation.	255.48	3.40	258.88	304 cu. yd.	0.85
1	" " —foundation.	555.53	1,331.03	1,886.56	173.8 cu. yd.	10.85
8405	" " and shells	821.67	22,238.28	23,059.95	162.53 tons	141.88
.01	Repairs to No. 2 stand	164.44	9.53	173.97	.....	.....
.1	Converter shells—brick lining	785.61	8,043.00	8,828.61	4 shells	2,207.15
.11	" " —unloading brick.	104.22	82.03	186.25	579.30 tons	0.32
8406	Cranes	1,438.50	23,027.65	24,466.15	110.75 tons	220.91
.1	" wiring	1,941.44	342.62	2,284.06	2 cranes	1,142.03
8407	Clinkering machines	1,715.23	13,981.94	15,697.17	2 machines	7,640.71
.01	" " alteration No. 1	33.00	1.43	34.43	.....	.....
.02	" " alteration No. 2.	57.92	40.39	98.31	.....	.....
.03	Clinkering machine electrical alteration	65.98	31.10	97.08	.....	.....
.1	Clinkering machines, wiring	392.21	283.79	676.00	2 machines	338.00
8409	Wiring for converter control	136.42	318.49	454.91	3 converters	151.64
.1	Lighting	451.93	462.01	913.94	60 drops	15.23
8410	Air pipe from power house excav	224.06	.....	224.06	331 cu. yd.	0.68
.1	Air pipe from power house laying	674.62	2,041.89	2,716.51	422 ft.	6.43
8411	Ladles, boats, bales, tools, etc	906.82	4,932.39	5,839.21	.....	.....

8400.—Converter Plant (*Continued*)

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8413	Casting machine excavation...	490.39	30.43	520.82	512 cu. yd.	1.02
8414	" " foundation...	1,627.37	1,889.36	3,516.73	291.9 cu. yd.	12.05
8415	" " cost and erection.	3,266.34	24,211.21	27,477.55	2 machines	13,738.78
.1	" " repairs..	221.71	13.88	235.59	.....	.....
8416	Loading platform excavation.	212.49	0.28	212.77	216 cu. yd.	0.99
.1	" " foundation...	311.35	505.68	817.03	93.2 cu. yd.	8.77
.11	" " floors...	381.02	1,053.55	1,434.57	6,803 sq. ft.	0.21
.2	" " backfill.	67.85	.....	67.85	129 cu. yd.	0.53
.3	" " striking plates..	53.09	126.69	179.78	119 sq. ft.	1.51
8417	Hoods and smoke boxes..	2,674.30	4,012.59	6,686.89	3 sets	2,228.96
.1	Hood to protect converter operator.....	62.67	109.66	172.33	1 hood	172.33
8418	Spouts, gates and hoppers at silica ore bins.....	245.48	1,400.37	1,645.85	.....	.....
8419.1	10-ton bullion scales excavation.....	19.76	.....	19.76	24 cu. yd.	0.82
.2	" " " foundation.....	58.32	65.19	123.51	10.5 cu. yd.	11.76
.3	" " " cost and erection.....	55.55	736.53	792.08	10 tons	79.21
.4	" " " scale house.....	48.78	41.09	89.87	1 shed	89.87
8425	Conveyor No. 15.....	422.77	2,251.47	2,674.24	165 ft.	16.21
8426.1	Wet pan excavation.....	2.44	.....	2.44	3 cu. yd.	0.81
.2	" " foundation.....	53.30	55.96	109.26	7.5 cu. yd.	14.57
.3	" " cost and erection.	304.76	1,050.10	1,354.86	1 pan	1,354.86
.4	" " bin and spout...	75.64	180.47	256.11	548 cwt.	4.67
	Total cost Converter plant	.....	.....	$216,033.37		

8420.—Converter Dust Chamber

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8421	Excavation.....	$127.28	.....	$127.28	265 cu. yd.	$0.48
8422	Foundation.....	668.85	$1,239.17	1,908.02	286.4 cu. yd.	6.66
8423	Steel structure.....	.....	.....	20,371.20	238.30 tons	85.49
.01	Wire baffles.....	138.86	1,101.95	1,240.81	166.10 c. wire	7.47
.1	Tile work.....	1,620.05	2,182.40	3,802.45	6,369 cu. ft.	0.60
.11	Tile handling.....	37.61	.....	37.61	155.20 tons	0.24
8424	Iron doors and frames....	1.94	158.93	160.87	.....	.....
8428	Smoke box track.....	155.32	10.02	165.34	.....	.....
	Total cost Converter dust chamber.....	.....	.....	$27,813.58		

8500.—Conveying System

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8501	Excavation	$1,824.22	.....	$1,824.22	2,286 cu. yd.	$0.80
8502	Foundation	2,550.78	$4,006.37	6,557.15	622.3 cu. yd.	10.54
8503	Steel structure	.....	.....	19,365.98	211.73 tons	91.47
8504	Woodwork	589.51	630.42	1,219.93	13.86 m.b.m.	88.02
.1	Floor battens	146.29	51.47	197.76	.....	.....
8505	Conveyors Nos. 3–4–5–6 11–14	1,210.87	12,505.62	13,716.49	1,284.9 ft.	10.68
.1	Chutes	665.15	167.13	832.28	23.3 cwt.	3.57
.2	Guides	28.00	.....	28.00	.....	.....
.3	Weightometer	65.00	1,329.92	1,394.92	1 weightometer	1,394.92
8506	Lighting	189.86	84.56	274.42	33 drops	8.32
	Total cost—Conveying system	.....	.....	$45,411.15		

8600.—Chimney

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8601	Excavation	$337.44	$29.61	$367.05	597 cu. yd.	$0.61
8602	Foundation	654.42	4,199.65	4,854.07	872.7 cu. yd.	5.56
8603	Brickwork	891.88	39,358.34	40,250.22	58,644 cu. ft.	0.69
	Total cost—Chimney	.....	.....	$45,471.34		

8610.—Reverberatory Flue

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8611	Excavation	$916.72	$20.21	$936.93	1,588 cu. yd.	$0.59
8612	Foundation	1,657.09	3,886.80	5,543.89	487.8 cu. yd.	11.37
8613	Brickwork	1,272.86	1,696.39	2,969.25	6,400 cu. ft.	0.46
.01	Unloading brick	57.79	.....	57.79	278.33 tons	0.21
8614	Steel structure	.....	.....	3,593.06	41.61 tons	86.35
.1	Clean out doors	10.61	153.61	164.22	.....	.....
.2	Caulking roof	184.35	4.21	188.56	.....	.....
	Total cost—Reverberatory flue	.....	.....	$13,453.70		

8620.—Converter Flue

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8621	Excavation	$168.02	.....	$168.02	198 cu. yd.	$0.85
8622	Foundation	165.73	$652.69	818.42	142 cu. yd.	5.76
8624	Steel structure	.....	.....	6,616.44	81.99 tons	80.70
	Total cost—Converter flue	.....	.....	$7,602.88		

8625.—Roaster Dust Chamber Flue

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8626	Excavation	$225.37	.....	$225.37	213 cu. yd.	$1.06
8627	Foundation	224.35	551.23	775.58	114.6 cu. yd.	6.77
8628	Brickwork	1,018.13	1,573.06	2,591.19	4,231 cu. ft.	0.61
.01	Unloading brick	57.60	.....	57.60	171.40 tons	0.34
8629	Steel structure	.....	.....	9,209.36	94.46 tons	97.49
	Total cost—Roaster dust chamber flue	.....	.....	$12,859.10		

8700.—Boiler and Blacksmith Shop

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8701	Excavation	$1,142.07	$44.81	$1,186.88	1,458 cu. yd.	$0.81
8702	Foundation	416.57	584.49	1,001.06	78.7 cu. yd.	12.71
8703	Steel structure	.....	.....	2,913.90	32.72 tons	89.06
.1	Doors, windows and frames	693.02	2,456.28	3,149.30	2,581 sq. ft. opening	1.22
.11	Concrete sills	119.80	50.87	170.67	251.5 lin. ft.	0.68
.2	Tile walls	477.95	612.62	1,090.57	2,297 cu. ft.	0.48
.21	Unloading tile	18.89	.....	18.89	69.70 tons	0.27
.22	Coping	112.17	2.72	114.89	290 lin. ft.	0.40
.30	Roof	286.52	828.24	1,114.76	66.49 squares	16.77
.31	Ventilators	16.01	261.50	277.51	3 vents	92.50
.40	Dirt floor	59.73	1.25	60.98	.....	.....
.50	Benches	87.83	49.43	137.26	.....	.....
.60	Painting	92.53	60.73	153.26	1,574 sq. yd.	0.10
8704	Crane	119.60	438.41	558.01	1 crane	558.01
8705	Tools	796.51	7,859.36	8,655.87	.....	.....
8706	Shafting, pulleys, belting	105.59	301.16	406.75	51 lin. ft.	7.98
8707	Motor	23.22	347.54	370.76	20 h.p.	18.54
8708	Lighting	23.41	44.50	67.91	17 drops	4.00
	Total cost—Boiler and blacksmith shop			$21,449.23		

8714.—Machine and Carpenter Shop

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8715	Excavation	$1,615.83	$325.28	$1,941.11	1,765 cu. yd.	$1.10
8716	Foundation	792.05	584.06	1,376.11	105.5 cu. yd.	13.04
8717	Steel structure	.....	.....	3,431.42	38.23 tons	89.76
.1	Doors, windows and frames	923.61	2,992.16	3,915.77	3,037 sq. ft. opening	1.29
.11	Concrete sills	111.65	67.70	179.35	295.3 lin. ft.	0.61
.2	Tile walls	531.45	571.28	1,102.73	2,397 cu. ft.	0.46
.21	Unloading tile	42.06	2.00	44.06	58.80 tons	0.75
.22	Wall coping	121.67	23.70	145.37	320 lin. ft.	0.45
.30	Roof	297.85	953.04	1,250.89	77.21 squares	16.20
.31	Ventilators	11.16	248.24	259.40	3 ventilators	86.45
.40	Floor	269.80	593.30	863.10	4,136 sq. ft.	0.21
.50	Benches	130.00	35.00	165.00	.....	.....
.60	Painting	118.00	87.40	205.40	1,989 sq. yd.	0.10
8718	Crane	25.19	564.36	589.55	.....	.....
8719	Tools	444.07	8,953.13	9,397.20	.....	.....
8720	Shafting, pulleys and belting	289.29	1,513.36	1,802.65	152 lin. ft.	11.86
8721	Motor	18.34	477.97	496.31	40 h.p.	12.40
8722	Lighting	55.84	135.01	190.85	20 drops	9.54
	Total cost—Machine and carpenter shop			$27,356.27		

8800.—General Office

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8804	Furniture and fixtures	.....	$1,394.95	$1,394.95	.....	.....

8809.—Warehouse

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8810	Excavation	$944.59	$51.49	$996.08	1,287 cu. yd.	$0.77
8811	Foundation	878.16	856.09	1,734.25	123 cu. yd.	14.09
8812	Steel structure	.....	.....	3,734.08	39.76 tons	93.92
.1	Doors, windows and frames	533.02	1,056.31	1,589.33	1,982 sq. ft. opening	0.80
.11	Concrete sills	164.72	61.63	226.35	241.5 lin. ft.	0.94
.2	Tile walls	438.00	477.86	915.86	2,342 cu. ft.	0.39
.21	Unloading tile	15.50	1.00	16.50	74.20 tons	0.22
.22	Coping	176.60	36.53	213.13	320 lin. ft.	0.67
.3	Painting roof	81.16	65.66	146.82	813 sq. ft.	0.18
.31	Ventilators	30.38	207.12	237.50	3 vents	79.17
.4	Floor excavation	129.03	.....	129.03	66 cu. yd.	1.96
.41	" concrete	558.04	721.60	1,279.64	8,298 sq. ft.	0.15
.5	Lighting	45.09	70.48	115.57	26 drops	4.45
8813	Warehouse fixtures	548.66	1,541.12	2,089.78	.....	.....
.1	Painting	26.50	14.17	40.67	412 sq. yd.	0.10
.11	" sash	122.78	15.34	138.12	189 sash	0.73
	Total cost—Warehouse			$13,602.71		

8819.—Laboratory

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8820	Excavation	$191.11	$0.16	$191.27	212 cu. yd.	$0.90
8821	Foundation	448.42	575.13	1,023.55	96.5 cu. yd.	10.61
.1	Plain concrete floors	113.78	154.90	268.68	1,026 sq. ft.	0.26
.2	Reinforced floors	59.81	114.87	174.68	364 sq. ft.	0.48
.3	Sills and lintels	109.53	22.85	132.38	163 lin. ft.	0.81
8822	Tile walls	250.71	517.66	768.37	841 cu. ft.	0.91
.2	Carpenter work	194.43	338.16	532.59	.....	.....
.5	Doors, windows and skylights	208.70	480.33	689.03	823 sq. ft. opening	0.84
8824	Wood fixtures	486.90	204.02	690.92	.....	.....
8825	Lighting	226.77	61.28	288.05	.....	.....
8826	Plumbing	129.79	97.23	227.02	.....	.....
8828	Painting	151.46	55.05	206.51	.....	.....
8829	Plastering	58.11	30.55	88.66	171.5 sq. yd.	0.52
8830	Apparatus	44.82	618.98	663.80	.....	.....
8831	Oil centrifuge	40.92	157.59	198.51	.....	.....
	Total cost—Laboratory			$6,144.02		

8840.—Sample Room

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8841	Excavation	$61.35	.....	$61.35	72 cu. yd.	$0.85
8842	Foundation	61.00	$64.43	125.43	9 cu. yd.	13.94
.1	Concrete floors	63.17	75.25	138.42	489 sq. ft.	0.28
8843	Walls and roof structure	101.84	201.81	303.65	.....	.....
.4	Roof	23.00	76.57	99.57	8 squares	12.45
.5	Doors and windows	32.39	118.85	151.24	298 sq. ft. opening	0.51
8844	Oven	233.74	58.56	292.30	.....	.....
8845	Benches, motor platform and fixtures	128.35	129.60	257.95	.....	.....
8846	Lighting	22.97	52.83	75.80	7 drops	10.82
8848	Painting	28.00	8.00	36.00	26 sash	1.38
8849	Machinery	53.07	912.00	965.07	.....	.....
.1	Motor	9.00	116.77	125.77	5 h.p.	25.15
.2	Shafting, pulleys and belting	23.39	170.17	193.56	26 lin. ft.	7.44
	Total cost—Sample room			$2,826.11		

8900.—Miscellaneous Accounts

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
8902	Sewer system, cost of pipe and laying	$778.83	$1,224.72	$2,003.55	2,967 ft.	$0.68
.1	" excavation	2,122.84	65.20	2,188.04	2,967 ft.	0.74
.2	" concrete	168.18	184.08	352.26	53.8 cu. yd.	6.55
8903	Outside closets	879.05	227.78	1,106.83	3 closets	368.94
8905	Permanent outside lighting	183.02	177.99	361.01	5 arcs	72.20
8906	Water pipe lines excavation	868.11	.....	868.11	4,253 ft.	0.20
.01	" concrete	17.37	17.86	35.23	2.3 cu. yd.	15.32
.02	" cost and laying	2,863.32	2,062.07	4,925.39	4,253 ft.	1.16
.1	6-in. pipe line to Clifton	1,474.71	6,914.95	8,389.66	8,988 ft.	0.93
.2	Water supply tank, excavation.	143.68	.....	143.68	116 cu. yd.	1.24
.4	" cost and erection	.....	.....	4,137.03	33.67 tons	122.87
8908	Power distribution	3,233.02	7,407.21	10,640.23	17,370 ft.	0.61
8909	Permanent air line, excavation	267.50	.....	267.50	401 cu. yd.	0.67
.1	" laying	432.37	623.08	1,055.45	2,316 ft.	0.46
8961	Steam heating system, excavation	166.36	.....	166.36	225 cu. yd.	0.73
.1	" cost and installation	240.78	305.37	546.15	496 ft.	1.10
	Total cost—Miscellaneous accounts			$37,186.48		

8999.—Charges to Indirect Expense

Number	Name of Account	Total
7001	General expense at Clifton	$13.58
7004	Personal injuries	6,734.01
8901	Derricks and construction equipment	18,718.35
8904	Telephone system	3,229.40
8905.1	Temporary outside lighting	18.30
8907	Watchman	1,516.69
8908.1	Temporary oil tanks	382.75
8910	Transmission of power to various departments	.....
8911	Watchman's house	185.39
8912	Tool shed	725.75
8913	Barn and corral	2,100.53
8914	Temporary blacksmith shop	251.71
8916	" power plant	5,518.11
8917	" crushing plant	4,555.07
8918	" water tank	1,991.31
8919	" electrical shop equipment	322.16
8920	Wagon roads	1,215.33
8921	Temporary pumping plant	375.25
8922	" pipe lines	5,199.73
8923	" warehouse	1,810.24
8924	" cement sheds	801.10
8925	Horses, harness and carts.	2,646.40
8926	Temporary office	1,195.86
8927	" closets	146.65
8928	" machine shop	166.10
8929	Employees' railroad transportation	16,994.57
8930	Clearing land	456.67
8931	Test holes	109.48
8933	Furniture and fixtures	365.25
8934	Miscellaneous supplies	4,436.24
8935	Shop equipment	657.68
8936	Overhead shop expense	8,394.19
8937	Stock lumber	.....
8938	Powder magazine	241.37
8939	Miscellaneous labor	6,825.89
8941	Temporary railway receiving bins	243.78
8942	Water supply	2,372.63
8943	Corral expense	.....
8944	Switching and freight from Clifton	2,661.35
8945	Office stationery and supplies	1,369.68
8946	Warehouse operating expense	10,771.88
8947	Time-keeping expense	4,346.59
8948	Form lumber	.....
8949	Cement	.....
8951	Sand and gravel	.....
8952	Employees' quarters	2,192.98
8953	Crushing plant operating expense	.....
8954	Concrete, power and repairs	.....
8955	Mortar sand	.....
8955.1	Mortar lime	.....
8955.2	Mortar cement	.....
8955.4	Fire brick mortar	.....
8955.5	Silica brick mortar	.....
8956	Operating temporary power house	.....
8957	Maintenance of tracks in yards	2,800.31
8958	Ditch at tunnel No. 2	5,851.89
8975	Cleaning up	4,996.20
8976	Rehandling brick and tile.	53.02
8998	Direct charges	4,316.30
8999	Total charges to indirect expense	$140,277.72
	Total unit cost	7.53 per cent.

This percentage is obtained by dividing the total charges to indirect expense by the total cost of the smelter, minus engineering and indirect expense.

$$\frac{\$140,277.72}{1,864,092.47} = 7.53 \text{ per cent.}$$

9000.—Power Plant

Power House

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
9000.1	Power plant engineering	.....	.....	.....	.....	See page 5
9001	Excavation.....	$7,727.56	$69.09	$7,796.65	7,313 cu. yd.	$1.07
9002	Bldg. foundation piers.	1,699.92	1,460.02	3,159.94	231.7 cu. yd.	13.64
.1	" " walls	3,735.78	3,628.81	7,364.59	508.5 cu. yd.	14.48
.2	North tunnel.....	1,350.79	1,230.37	2,581.36	180.3 cu. yd.	14.32
.3	Concrete drain.....	205.68	227.37	433.05	34.6 cu. yd.	12.52
.4	Basement floor, concrete.....	916.41	1,347.78	2,264.19	12,130 sq. ft.	0.19
.45	" painting.....	81.45	48.81	130.26	830 sq. yd.	0.16
.6	Preparation of concrete for painting.....	891.73	42.69	934.42	2,459 sq. yd.	0.38
.7	Painting concrete.....	195.84	301.61	497.45	2,459 sq. yd.	0.20
9003	Steel structure.....	.....	.....	23,773.30	254.29 tons	93.49
.1	Tile walls.....	3,856.83	4,510.20	8,367.03	14,343 cu. ft.	0.58
.11	Unloading tile.....	332.40	0.17	332.57	522.70 tons	0.64
.12	Wall coping.....	372.69	107.05	479.74	732 lin. ft.	0.66
.2	Doors, window and frames.....	974.38	3,319.93	4,294.31	4,044 sq. ft. opening	1.06
.21	Concrete sills.....	596.33	120.96	717.29	964 lin. ft.	0.74
.3	Ventilators.....	125.60	439.76	565.36	6 ventilators	94.23
.4	Main floor columns...	236.93	626.44	863.37	68 columns	12.70
.41	Main floor slab concrete.....	1,267.91	3,341.61	4,609.52	10,210 sq. ft.	0.45
.42	Painting under side main floor.....	181.88	147.58	329.46	2,679 sq. yd.	0.12
.43	" top main floor...	95.56	199.32	294.88	1,134 sq. yd.	0.26
.5	Roof, Berger multiplex plate.....	420.83	3,063.18	3,484.01	214.83 squares	16 22
.51	" concrete.....	1,723.10	958.51	2,681.61	214.83 "	12.48
.52	" tar.....	172.70	127.73	300.43	214.83 "	1.40
.53	" down spouts and tile drain.....	286.17	240.44	526.61	905 ft.	0.58
.54	" painting underside.....	692.84	324.55	1,017.39	6,813 sq. yd.	0.15
.55	" P. & B. roofing..	577.68	1,317.08	1,894.76	214.83 squares	8.82
.60	Painting sash.....	290.09	16.72	306.81	299 sash	1.02
.61	Painting woodwork...	29.50	4.06	33.56	89 sq. yd.	0.38
9004	Crane.....	131.89	1,723.27	1,855.16	1 crane	1,855.16
9005	Well grading.....	1,558.07	517.68	2,075.75	2,600 cu. yd.	0.80
.1	Shaft sinking.....	765.62	612.10	1,377.72	45 ft.	30.61
.2	Timbering.....	57.61	.....	57.61	45 ft.	1.28
.31	Aldrich pump installation.....	74.56	16.62	91.18	.....	.....
9006.01	Nordberg blowers, foundation.....	774.06	3,020.83	3,794.89	686.3 cu. yd.	5.53
.1	" cost and installation.....	1,641.62	32,514.02	34,155.64	2 Nordbergs	17,077.82
.2	" painting.....	327.57	57.65	385.22	2 "	192.61
9007.01	Turbines, foundation..	959.08	1,432.70	2,391.78	196.5 cu. yd.	12.16
.1	" cost and installation.....	2,297.70	79,586.49	81,884.19	3 turbines	27,294.73
.2	" painting.....	286.15	41.02	327.17	3 "	109.06
.3	" air pipe making..	547.68	200.75	748.43	103 ft.	6.27
.4	" air pipe erection.	232.57	64.24	296.81	103 ft.	2.88
.51	Transformer trucks and transfer table...	121.63	538.08	659.71	15 trucks	43.98
.52	Auto transformers....	735.60	12,044.91	12,780.51	10 transformers	1,278.05
9008.01	Condensers foundations.....	291.08	285.18	576.26	50.3 cu. yd.	11.45

9000.—Power Plant (*Continued*)

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
9008 1	Condensers cost and installation	415.31	19,563.55	19,978.86	3 condensers	6,659.62
.2	" painting	30.00	5.86	35.86	3 "	11.95
9009	Jet condenser hot well, excavation	28.82	0.90	29.72	46 cu. yd.	0.65
.01	" foundation	66.27	69.99	136.26	16.5 "	8.26
.02	" supporting struct. and tank	.....	.....	945.74	5.76 tons	164.19
.03	" cost and erection	128.97	949.68	1,078.65	1 condenser	1,078.65
.12	" dry vacuum pumps	285.51	2,860.01	3,145.52	2 pumps	1,572.76
.13	" " painting	30.00	5.86	35.86	2 "	17.93
.21	Circulating pumps foundation	560.04	708.93	1,268.97	210 cu. yd.	6.04
.22	" cost and erection	366.90	3,535.68	3,902.58	2 pumps	1,951.29
.23	" painting	30.00	5.86	35.86	2 "	17.93
9010.01	Air compressor foundation	840.98	1,246.54	2,087.52	238.3 cu. yd.	8.76
.02	" erection	642.90	148.67	791.57	.....	.....
.03	" painting	10.58	24.49	35.07	.....	.....
.04	" all piping except steam	298.46	160.65	459.11	.....	.....
.05	" wrecking and transportation	457.77	136.06	593.83	.....	.....
.06	" installation of air receivers	49.47	1.43	50.90	.....	.....
9011.01	2 exciters, 2 air pumps, 2 dir. pumps, found'n	1,439.67	1,875.43	3,315.10	373 cu. yd.	8.89
.02	2 exciters, cost and installation	491.01	6,118.26	6,609.27	2 exciters	3,304.64
.03	3 dry vacuum pumps, cost and installation	147.26	3,190.10	3,337.36	3 pumps	1,112.45
.04	3 dir. pumps and engines, cost and installation	389.32	8,729 37	9,118.69	3 "	3,039.56
.05	2 exciters, painting	86.01	14.65	100.66	2 exciters	50.33
.06	3 air pumps, "	50.00	8.79	58.79	3 pumps	19.59
.07	3 dir. " "	81.69	14.65	96.34	3 "	32.11
9012.01	2 motor gen., 1 air pump, 1 dir. pump., foundation	296.52	658.91	955.43	107 cu. yd.	8.93
.02	2 motor generators, cost and installation	319.06	6,830.33	7,149.39	2 generators	3,574.69
.05	" painting	30.00	5.86	35.86	2 "	17.93
9013	Transfer table pit, concrete	24.13	58.23	82.36	12 cu. yd.	6.86
.01	Switchboard concrete compartments	1,472.21	510.48	1,982.69	1,469 sq. ft.	1.35
.02	" cost and erection	2,730.53	15,520.57	18,251.10	.....	.....
9014	Steam piping north and south mains, excavation	249.65	.....	249.65	279 cu. yd.	0.89
.01	" " foundation	578.24	945.97	1,524.21	194.5 cu. yd.	7.84
.02	" steel supporting structure	.....	.....	7,694.58	86.81 tons	88.64
.03	" hangers and anchors	1,030.68	337.26	1,367.94	153 rods	8.94
.04	" cost and erection	2,286.31	18,622.25	20,908.56	3,401 ft.	6.15
.05	" covering and erection	266.71	5,813.23	6,079.94	3,401 ft.	1.79

9000.—Power Plant (*Continued*)

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
9015	Exhaust pipe, cost and erection	1,745.71	8,715.66	10,461.37	1,541 ft.	6.79
.01	" painting	85.05	51.19	136.24	1,541 ft.	0.09
.05	" covering and erection	318.25	830.56	1,148.81	746 ft.	1.54
.10	Air piping, cost and erection	363.19	554.16	917.35		
.11	" painting	31.56	18.66	50.22		
.2	Exhaust pipe foundation	63.09	102.81	165.90	18.3 cu. yd.	9.07
.21	" supporting structure	197.27	57.93	255.20		
.22	" excavation	20.82		20.82	29 cu. yd.	0.72
9016	Water pipe, excavation and backfill	1,485.10	0.24	1,485.34	2,406 cu. yd.	0.62
.01	" cost and erection	3,747.79	16,437.88	20,185.67		
.02	" painting	230.59	25.54	256.13		
	Feed-water Heating Plant					
9017	Excavation	239.39	1.70	241.09	274 cu. yd.	10.88
.01	Foundation	708.06	551.04	1,259.10	105.4 cu. yd.	1.95
.015	Reinforced floors	282.46	273.51	555.97	1,330 sq. ft.	0.42
.016	Water-proofing concrete tanks	80.65	14.46	95.11	1,475 sq. ft.	80.06
.02	Steel structure			2,262.90	26.63 tons	4.98
.021	Distributing and equalising tank	364.70	260.58	625.28	80.2 cwt.	7.80
.03	Tile work	285.99	234.36	520.35	706 cu. ft.	0.74
.031	Unloading tile	24.91		24.91	30 tons	0.83
.032	Coping	14.83	14.01	28.84	108 lin. ft.	0.27
.033	Sills and lintels	7.81	5.57	13.38	60 lin. ft.	0.22
.034	Painting tile walls	26.10		26.10	112 sq. yd.	0.23
.035	Doors, windows and frames	59.83	99.98	159.81	186 sq. ft. opening	0.86
.04	Roofing	115.25	109.88	225.13	8.8 squares	25.58
.045	Ventilators	142.14	49.20	191.34	2 ventilators	95.67
.05	Treating tank, concrete	637.67	487.19	1,124.86	28.7 cu. yd.	39.19
.06	Receiving tank No. 1.	251.55	167.92	419.47	7.3 cu. yd.	57.46
.07	Receiving tank No. 2.	364.50	265.08	629.58	24.4 cu. yd.	25.80
.08	Calibrating tank	188.96	50.21	239.17	12.6 cwt.	18.98
.081	Tipping meter	372.46	227.50	599.96	1 tipping meter	599.96
.09	Heaters, recorder	685.22	2,813.84	3,499.06		
.1	Sewer excavation and backfill	157.19		157.19	266 cu. yd.	0.59
.11	Sewer pipe, cost and laying	71.88	203.00	274.88	100 ft.	2.75
.12	Lighting	53.35	25.83	79.18	6 drops	13.20
.13	Painting	62.78	20.35	83.13		
.14	Wood walkway and tank covers	67.08	51.31	118.39	1.56 m.b.m.	75.89
.15	Alterations	99.70	3.40	103.10		
	Condensed Water Pump House					
.20	Excavation	220.56	8.93	229.49	236 cu. yd.	0.97
.21	Foundation	1,171.89	854.68	2,026.57	171 cu. yd.	11.85
.22	Floor	78.41	57.70	136.11	355 sq. ft.	0.38
.24	Doors, windows and frames	26.02	22.68	48.70	57 sq. ft. opening	0.85
242	Tile work	98.87	84.28	183.15	257 cu. ft.	0.71
.243	Coping	14.50	2.73	17.23	57 lin. ft.	0.30
.25	Roof	73.92	60.83	134.75	5 squares	26.95
.26	Pumps and piping	90.58	691.37	781.95	2 pumps	390.98
.27	Lighting	221.16	51.53	272.69		

Power House Miscellaneous Accounts

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
9018.1	Power and lighting transformers	397.70	4,714.33	5,112.03	7 transformers	730.29
9019	Lighting	307.86	1,449.97	1,757.83	94 drops	18.70
9020	Oiling system	1,051.82	401.69	1,453.51	104 outlets	13.98
9021	Benches, bolt racks, etc.	161.22	51.80	213.02	.....	.....
9022	Instruments and gauges	57.97	343.06	401.03	.....	.....

Cooling Tower

9050	Excavation	1,590.85	83.47	1,674.32	1,589 cu. yd.	1.05
.01	Backfill	1,395.20	.....	1,395.20	2,415 cu. yd.	0.58
9051	Foundations, sumps and gutters	5,212.31	5,640.13	10,852.44	706.3 cu. yd.	15.37
.02	Floor	911.93	1,985.03	2,896.96	17,116 sq. ft.	0.17
.03	Water proofing concrete	588.76	130.59	719.35	1,606 sq. yd.	0.45
9052	Woodwork	3,181.33	5,415.47	8,596.80	128.63 m.b.m.	66.84
9053	Alterations	115.40	22.54	137.94	.....	.....
	Total cost—Cooling tower			$26,273.01		
	Total cost—Power plant			$434,703.15		

9000.—Oil Supply Sump and Pump House

Number	Name of Account	Labor	Material	Total	Quantity	Total Unit Cost
9060	Excavation	$1,148.78	$108.24	$1,257.02	1,308 cu. yd.	$0.96
.01	Concrete work	2,338.72	3,230.46	5,569.18	340.6 cu. yd.	16.35
.02	Pumps	176.20	2,035.58	2,211.78	2 pumps	1,105.89
.03	Inlet piping	44.77	126.55	171.32	108 ft.	1.59
.04	Lighting	79.71	62.57	142.28	4 drops	35.57
.05	Steel work	96.94	129.39	226.33	1.12 tons	202.08
.06	Doors, windows and frames	83.60	65.02	148.62	124 sq. ft. opening	1.20
.07	Roof	136.81	162.66	299.47	6.5 squares	46.07
.075	Ventilators	95.75	64.32	160.07	2 ventilators	80.03

Two 500,000-Gallon Oil Tanks

9060.10	Wrecking and transportation	934.00	465.31	1,399.31	64.40 tons	21.73
.11	Excavation	308.20	.....	308.20	554 cu. yd.	0.56
.12	Foundation	128.70	210.61	339.31	32.8 cu. yd.	10.35
.13	Erection	3,602.89	429.47	4,032.36	64.40 tons	62.62
.131	Roof supports	362.90	359.83	722.73	65.56 squares	11.02
.132	Sheathing, lath and plaster	399.05	523.01	922.06	75.50 squares	12.22

Tracks at Oil Sump

9060.14	Railroad grading	1,477.27	.....	1,477.27	2,439 cu. yd.	0.61
.15	Laying and ballasting	1,107.38	1,092.99	2,200.37	1,362 ft.	1.62
.16	Track bumpers	246.74	47.92	294.66	3.16 m.b.m.	93.25
.17	Bridges over wood pipe	229.22	87.14	316.36	.....	.....

Oil Supply Tanks for Reverberatories and Boilers

9060.20	Excavation	392.13	13.18	405.31	404 cu. yd.	1.00
.21	Foundation	875.70	1,685.76	2,561.46	189.5 cu. yd.	13.52
.22	Cost and erection	.....	.....	3,926.67	8 tanks	487.94
.23	Piping	199.41	282.16	481.57	785 ft.	0.61

Oil Piping

9060.40	Excavation	990.73	1.39	992.12	1,150 cu. yd.	0.86
.41	Pipe and laying	3,156.14	5,654.50	8,810.64	1,888 ft.	4.67
9060.50	Heating installation	167.37	1,068.04	1,235.41	360 ft.	3.43
	Total cost—Oil supply system			$40,611.88		

NEW SMELTER

Recapitulation of Costs

Number	Name of Account	Total	Number	Name of Account	Total
7100	Engineering expense....	$100,649.88	8625	Roaster dust chamber flue.....	12,859.10
7300	Yard tracks and industrial system.....	156,326.43	8700	Boiler and blacksmith shop.....	21,449.23
7400	Receiving bins.....	44,185.06	8714	Machine and carpenter shop.....	27,356.27
7700	Crushing plant.....	9,268.62	8800	General office.....	1,394.95
7800	Sampling plant.....	34,108.74	8809	Warehouse.....	13,602.71
7900	Bedding plant and bunker bins.....	150,939.05	8819	Laboratory.....	6,144.02
8100	Roasting plant.....	136,734.87	8840	Sample room.....	2,826.11
8120	Roaster dust chamber..	49,664.76	8900	Miscellaneous accounts.	37,186.48
8300	Reverberatory plant....	328,945.02	8999	Indirect expense.....	140,277.72
8400	Converter plant.....	216,033.37	9000	Power plant.....	434,703.15
8420	Converter dust chamber	27,813.58	9060	Oil supply sump and pump house.....	40,611.88
8500	Conveying system.....	45,411.15			
8600	Chimney.....	45,471.34			
8610	Reverberatory flue.....	13,453.70			
8620	Converter flue.....	7,602.88		Total cost.....	$2,105,020.07

CHAPTER II

COMPARATIVE COSTS

Plain Concrete

Number	Name of Account	Cu. Yd.	Labor Cost Amount	Labor Cost per Cu. Yd.	Material Cost Amount	Material Cost per Cu. Yd.	Total Cost Amount	Total Cost per Cu. Yd.
7308.1	Trestle approach to reverbs.....	254.9	$738.18	$2.90	$1,152.42	$4.52	$1,890.60	$7.42
7309.1	Track scales, receiving yard.....	186.0	545.62	2.93	1,146.87	6.17	1,692.49	9.10
7310	Bridge No. 1.....	339.8	1,028.26	3.03	1,968.38	5.79	2,996.64	8.82
7312.1	Retaining wall.....	203.5	512.34	2.52	734.25	3.61	1,246.59	6.13
7313.1	Calcine track scales..	41.6	207.55	4.99	193.60	4.65	401.15	9.64
7314.1	Approach to receiving bins.....	754.3	2,408.16	3.19	3,017.95	4.00	5,426.11	7.19
7402	Receiving bins	612.3	1,235.51	2.02	2,247.70	3.67	3,483.21	5.69
7802	Sampling plant.....	120.7	605.91	5.02	649.44	5.38	1,255.35	10.40
8102	Roaster plant.....	250.5	765.42	3.06	1,069.63	4.27	1,835.05	7.33
8122	Roaster dust chamber	472.9	1,049.71	2.22	1,775.05	3.75	2,824.76	5.97
8306.1	Flues from boilers to reverb. flue.....	9.2	74.22	8.07	74.02	8.04	148.24	16.11
8404.1	Converter stands....	173.8	555.53	3.19	1,331.03	7.66	1,886 56	10.85
8419.2	Bullion scales.....	10.5	58.32	5.55	65.19	6.21	123.51	11.76
8422	Converter dust chamber.....	286.4	668.85	2.33	1,239.17	4.33	1,908.02	6.66
8426.20	Wet pan.....	7.5	53.30	7.10	55.96	7.46	109.26	14.56
8602	Chimney.....	872.7	654.42	0.75	4,199.65	4.81	4,854.07	5.56
8622	Converter flue.....	142.0	165.73	1.17	652.69	4.59	818.42	5.76
8627	Roaster dust chamber flue.....	114.6	224.35	1.96	551.23	4.81	775.58	6.77
8702	Boiler and blacksmith shop.....	78.7	416.57	5.29	584.49	7.42	1,001.06	12.71
8716	Machine and carpenter shop.....	105.5	792.05	7.51	584.06	5.53	1,376.11	13.04
8811	Warehouse.....	123.0	878.16	7.14	856.09	6.96	1,734.25	14.09
8842	Sample room.....	9.0	61.00	6.78	64.43	7.16	125.43	13.94
8901	Derricks and cons. equipment.....	53.8	168.18	3.13	184.08	3.42	352.26	6.55

Plain Concrete (*Continued*)

Number	Name of Account	Cu. Yd.	Labor Cost Amount	Labor Cost per Cu. Yd.	Material Cost Amount	Material Cost per Cu. Yd.	Total Cost Amount	Total Cost per Cu. Yd.
8906.01	Water pipe lines, etc.	2.3	17.37	7.55	17.85	7.76	35.23	15.31
8908.2	Power conduits	27.6	198.14	7.18	217.15	7.87	415.29	15.04
8908.5	Power conduit branches	5.0	25.36	5.07	30.63	6.12	55.99	11.19
8916	Temporary power plant	14.4	56.04	3.89	98.69	6.86	154.73	10.75
8917	" crushing plant	101.1	303.15	3.00	534.13	5.28	837.28	8.28
8918	" oil and water tanks	48.8	149.27	3.06	263.01	5 39	412.28	8.45
9002	Power plant	231.7	1,699.92	7.34	1,460.02	6.30	3,159.94	13.64
9002.3	" drain	34.6	205.68	5.94	227.37	6.57	433.05	12.52
9006.01	" Nordberg engines	686.3	774.06	1.13	3,020.83	4.40	3,794.89	5.53
9007.01	" turbines	196.5	959.08	4.88	1,432.70	7.29	2,391.28	12.16
9008.01	" condensers	50.3	291.08	5.79	285.18	5.67	576.26	11.45
9009.01	" jet condenser	16.5	66.27	4.02	69.99	4.24	136.26	8.26
9009.21	" 2 air pumps	210.0	560.04	2.67	708.93	3.37	1,268.97	6.04
9010.01	" air compressor	238.3	840.98	3.53	1,246.54	5.23	2,087.52	8.76
9011.01	" exciters, etc.	373.0	1,439.67	3.86	1,875.43	5.03	3,315.10	8.89
9012.01	" motor generators	107.0	296.52	2.77	658.91	6.16	955.43	8.93
9014.01	North and south steam mains	194.5	578.24	2.97	945.97	4.86	1,524.21	7.84
9015.2	Power plant exhaust pipe	18.3	63.09	3.45	102.81	5.62	165.90	9.07
	Total	7,779.4	$22,391.30	$2.85	$37,593.53	$4.82	$59,984.83	$7.67

Miscellaneous Concrete

Number	Name of Account	Cu. Yd.	Labor Cost Amount	Labor Cost per Cu. Yd.	Material Cost Amount	Material Cost per Cu. Yd.	Total Cost Amount	Total Cost per Cu. Yd.
7702	Crusher plant	220.5	$893.15	$4.05	$1,568.47	$7.11	$2,461.62	$11.16
7902	Bedding plant	2,809.7	6,256.34	2.23	14,513.21	5.16	20,769.55	7.39
8302	Reverberatory plant	2,810.0	7,715.96	2.74	15,044.09	5.35	22,760.05	8.10
8307.11	Waste heat boilers	138.7	789.52	5.69	793.97	5.72	1,583.49	11.42
8307.12	Oil-fired boilers	100.0	350.45	3.50	686.29	6.86	1,036.74	10.37
8307.13	Feed pumps	214.7	678.73	3.16	1,428.97	6.66	2,107.70	9.82
8402.	Converter plant	776.9	1,796.56	2.31	2,860.16	3.68	4,656.72	6.00
8414	Casting machines	291.9	1,627.37	5.58	1,889.36	6.47	3,516.73	12.05
8502	Conveying system	622.3	2,550.78	4.10	4,006.37	6.44	6.557.15	10.54
8821	Laboratory	96.5	448.42	4.65	575.13	5.96	1,023.55	10.61
8902.2	Sewer system	53.8	168.18	3.13	184.08	3.42	352.26	6.55
9013	Transfer table pit	12.0	24.13	2.01	58.23	4.85	82.36	6.86
9017.01	Feed water heating plant	105.4	708.06	6.72	551.04	5.23	1,259.10	11.95
9017.21	Condensed water pump house	171.0	1,171.89	6.85	854.68	5.00	2,026.57	11.85
9060.21	Supply tanks, reverbs. and oil-fired boilers	189.5	875.70	4.62	1,685.76	8.90	2,561.46	13.52
	Total	8,706.1	$26,055.24	$2.99	$46,699.81	$5.36	$72,755.05	$8.36

Reinforced Concrete

Number	Name of Account	Cu. Yd.	Labor Cost Amount	Labor Cost per Cu. Yd.	Material Cost Amount	Material Cost per Cu. Yd.	Total Cost Amount	Total Cost per Cu. Yd.
8307.10	Reverb. boiler bldg.	573.7	$2,181.08	$3.80	$3,846.18	$6.70	$6,027.26	$10.51
8612	Reverberatory flue.	487.8	1,657.09	3.40	3,886.80	7.97	5,543.89	11.37
9002.1	Power house.....	508.5	3,735.78	7.35	3,628.81	7.14	7,364.59	14.48
.2	North tunnel.....	180.3	1,350.79	7.49	1,230.57	6.83	2,581.36	14.32
9051	Cooling tower.....	706.3	5,212.31	7.38	5,640.13	7.99	10,852.44	15.37
9060.01	Oil supply sump and pump house.	340.6	2,338.72	6.87	3,230.46	9.48	5,569.18	16.35
.12	Oil storage tanks..	32.8	128.70	3.92	210.61	6.42	339.31	10.35
	Total.....	2,830.0	$16,604.47	$5.87	$21,673.56	$7.66	$38,278.03	$13.53
	Total concrete foundations.....	19,315.5	$65,051.01	$3.37	$105,966.90	$5.48	$171,017.91	$8.85

Plain Concrete Floors

Number	Name of Account	Sq. Ft.	Thickness	Labor Cost Amount	Labor Cost per Sq. Ft.	Material Cost Amount	Material Cost per Sq. Ft.	Total Cost Amount	Total Cost per Sq. Ft.
7802.1	Sampling plant, ground floor.....	1,222	5 in.	$105.68	$0.09	$263.92	$0.21	$369.60	$0.30
8307.3	Around boilers.....	2,705	4 "	276.29	0.10	664.82	0.25	941.11	0.35
8812.41	Warehouse floor and platform....	8,298	4 "	558.04	0.07	721.60	0.09	1,279.64	0.15
8821.1	Laboratory.....	1,026	4 "	113.78	0.11	154.90	0.15	268.68	0.26
8842.1	Sample room.....	489	5 "	63.17	0.13	75.25	0.15	138.42	0.28
9002.4	Power plant basement.....	12,130	5 "	916.41	0.07	1,347.78	0.11	2,264.19	0.18
9017.22	Condensed water pump house.....	355	4 "	78.41	0.22	57.70	0.16	136.11	0.38
		26,225		$2,111.78	$0.08	$3,285.97	$0.13	$5,397.75	$0.21
	The above concrete floors were laid in small blocks 5 to 6 ft. square, having a finished top and with sand joints between all squares								
8416.11	Loading platform..	6,803	5 in.	381.02	0.06	1,053.55	0.15	1,434.57	0.21
	The floor above was finished as a large slab with no sand joints								

Reinforced Concrete Floors

Number	Name of Account	Sq. Ft.	Thickness	Labor Cost Amount	Labor Cost per Sq. Ft.	Material Cost Amount	Material Cost per Sq. Ft.	Total Cost Amount	Total Cost per Sq. Ft.
7802.2	Sampling plant....	4,244	4½ in.	$1,050.61	$0.25	$1,080.30	$0.25	$2,130.91	$0.50
8307.2	Over slag track cut	7,676	5 "	1,018.17	0.13	1,687.29	0.22	2,705.46	0.35
8821.1	Laboratory.....	364	4½ "	59.81	0.16	114.87	0.32	174.68	0.48
9017.015	Feed water heating plant.....	1,330	4½ "	282.46	0.21	273.51	0.21	555.97	0.42
		13,614		$2,411.05	$0.18	$3,155.97	$0.23	$5,567.02	$0.41
	The floors above were formed, reinforced and finished								
9003.41	Power plant main floor.....	10,210	3½ in.	1,267.91	0.12	3,341.61	0.33	4,609.52	0.45
	The floor above was laid on Berger plate, reinforced and finished								
9051.02	Power plant cooling tower.....	17,116	4 in.	911.93	0.05	1,985.03	0.12	2,896.96	0.17
	The floor above was reinforced, laid on the ground and straight-edged but with no finish								

Coping

Number	Name of Account	Lin. Ft.	Labor Cost Amount	Labor Cost per Ft.	Material Cost Amount	Material Cost per Ft.	Total Cost Amount	Total Cost per Ft.
8703.22	Boiler and blacksmith shop....	290	$112.17	$0.38	$2.72	$0.01	$114.89	$0.40
8717.22	Machine and carpenter shop...	320	121.67	0.37	23.70	0.08	145.37	0.45
8812.22	Warehouse.....	320	176.60	0.55	36.53	0.12	213.13	0.67
9017.243	Condensed water pump house..	57	14.50	0.25	2.73	0.04	17.23	0.30
		987	$424.94	$0.43	$65.68	$0.07	$490.62	$0.50
	This coping was 12 in. deep, projecting 2 in. from tile work							
9003.12	Power house.....	732	$372.69	$0.51	$107.05	$0.15	$479.74	$0.66
	This coping was 18 in. deep, projecting 2 in. from tile work							

Excavation—Type No. 1

This class covers shallow excavation made with picks, shovels, wheelbarrow and slips. The haul is less than 100 ft.

Number	Name of Account	Total Cost.	Cu. Yd.	Cost per Cu. Yd.
7308	Trestle approach to reverb. bldg.....	$359.95	277	$1.30
7313	Track scales on calcine track.....	108.95	118	0.92
7314	Trestles to receiving bins.....	548.18	589	0.93
7801	Sampling plant.....	295.06	332	0.89
8410	Air pipe line from power house.....	224.06	331	0.68
8416	Loading platform.....	212.77	216	0.99
8419.1	Bullion scales.....	19.76	24	0.82
8501	Conveying system.....	1,824.22	2,286	0.80
8611	Reverberatory flue.....	936.93	1,588	0.59
8621	Converter flue.....	168.02	198	0.85
8626	Roaster dust chamber flue.....	225.37	213	1.06
8812.4	Warehouse, floor.....	129.03	66	1.96
8820	Laboratory.....	191.27	212	0.90
8841	Sample room.....	61.35	72	0.85
8902.1	Sewer system.....	2,188.04	2,808	0.78
8906	Water pipe lines, tanks, etc.....	868.11	1,078	0.81
8908	Power conduit.....	358.53	435	0.82
8908	Power conduit branches.....	49.20	53	0.93
8909	Permanent air lines.....	267.50	401	0.67
8922	Temporary pipe lines.....	169.13	148	1.14
8923	Temporary warehouse.....	47.29	87	0.54
9009	Jet condenser hot well.....	29.72	46	0.65
9014	North and south steam mains.....	249.65	279	0.89
9015.22	Exhaust pipe.....	20.82	29	0.72
9017.20	Condensed water pump house.....	229.49	236	0.97
9060.40	Oil piping.....	992.12	1,150	0.86
	Total.....	$10,774.52	13,272	$0.81

Excavation—Type No. 2

This class covers excavation made with picks, shovels, slips and carts. The haul is over 100 ft. in every case.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
7309	Track scales in receiving yard	$348.91	388	$0.90
7312	Retaining walls	77.66	60	1.29
7701	Crushing plant	689.67	609	1.13
8306	Flues from boilers to reverbs	19.86	15	1.32
8307.01	Waste heat boilers	213.44	129	1.65
8307.03	Feed pumps	601.44	659	0.91
8404	Converter stands	258.88	304	0.85
8413	Two casting machines	520.82	512	1.02
8421	Converter dust chamber	127.28	265	0.48
8426.1	Wet pan	2.44	3	0.81
9017	Feed water heating plant	241.09	274	0.88
9050	Cooling tower	1,674.32	1,589	1.05
9060	Oil supply sump and pump house	1,257.02	1,308	0.96
9060.11	Storage tanks	308.20	554	0.56
	Total	$6,341.03	6,669	$0.95

Excavation—Type No. 3

This class covers excavation made with powder, picks, shovels and wheelbarrows. The haul was less than 100 ft.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
8906.2	Water supply tank	$143.68	116	$1.24
8916	Temporary power plant	354.07	388	0.91
8917	Temporary crushing plant	57.00	156	0.37
8920	Wagon road	924.74	951	0.97
9005	Well grading	2,075.75	2,600	0.80
	Total	$3,555.24	4,211	$0.84

Excavation—Type No. 4

This class covers excavation made with powder, picks, shovels, slips, fresnos and carts. The haul was over 100 ft.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
7901	Bedding plant	$12,259.43	12,319	$0.99
8101	Roaster plant	1,547.07	1,216	1.27
8307	Reverb. boiler building	283.14	306	0.93
8312.10	Feed piping from heating plant to pumps	1,091.42	1,296	0.84
9060.20	Supply tank for reverbs. and boilers	405.31	404	1.00
	Total	$15,586.37	15,541	$1.00

Excavation—Type No. 5

This class covers excavation done with plows, slips, fresnos, and in some cases powder. The haul was less than 100 ft.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
8701	Boiler and blacksmith shop	$1,186.88	1,458	$0.82
9001	Power house	7,796.65	7,313	1.07
9060.14	Railroad grading at oil sump	1,477.27	2,439	0.61
	Total	$10,460.80	11,210	$0.93

Excavation—Type No. 6

This class covers excavation made with plows, slips, fresnos, and in some cases powder. The haul was over 100 ft.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
8121	Roaster dust chamber	$1,112.83	1,194	$0.93
8301	Reverberatory plant	1,477.66	1,890	0.78
8307.02	Oil-fired boilers	73.60	97	0.76
8401	Converter plant	5,731.11	6,330	0.91
8601	Chimney	367.05	597	0.61
8715	Machine shop	1,941.11	1,765	1.10
8810	Warehouse	996.08	1,287	0.77
	Total	$11,699.44	13,160	$0.89

Excavation—Type No. 7

These are miscellaneous jobs where a variety of methods were used.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
7301	Yard tracks and industrial system	$35,566.00	55,405	$0.64
8901	Derricks and construction equip	30.32	41	0.74
8905.01	Permanent outside lighting	17.87	21	0.85
8952	Employes quarters	401.63	318	1.26
8961	Steam heating installation	166.36	228	0.73
9016	Power house water pipe	1,485.34	2,406	0.62
9017.1	Feed water heating plant	157.19	266	0.59
	Total	$37,824.71	58,685	$0.64

Excavation—Type No. 8

This class covers excavation made with picks, shovels, wheelbarrows and carts. A large portion of it was windlassed from deep pits.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
7401	Receiving bins	$2,342.27	1,428	$1.64

Excavation—Type No. 9

This covers backfilling and tamping in 4 to 5-in. layers.

Number	Name of Account	Total Cost	Cu. Yd.	Cost per Cu. Yd.
8301.01	Reverberatory plant	$2,755.55	3,679	$0.75
8307.04	Reverb. boiler building	548.10	972	0.56
8416.2	Loading platform	67.85	129	0.53
9050.01	Cooling tower	1,395.20	2,415	0.58
	Total	$4,766.70	7,195	$0.66
	Total excavation	$103,351.08	131,371	$0.79

Lighting

Number	Name of Account	No. Drops	Labor Cost Amount	Labor Cost per Drop	Material Cost Amount	Material Cost per Drop	Total Cost Amount	Total Cost per Drop
7407	Receiving bins	22	$60.87	$2.77	$24.67	$1.12	$85.54	$3.89
7707.1	Crushing plant	8	76.41	9.55	38.22	4.78	114.63	14.33
7806.1	Sampling plant	36	178.80	4.97	146.22	4.06	325.02	9.03
7906	Bedding plant	63	306.85	4.87	127.38	2.02	434.23	6.89
8109	Roaster plant	67	340.64	5.09	157.70	2.35	498.34	7.44
8313.1	Reverb. and boiler bldg.	104	612.30	5.89	473.19	4.55	1,085.49	10.44
8409.1	Converter plant	60	451.93	7.53	462.01	7.70	913.94	15.23
8506	Conveying system	33	189.86	5.76	84.56	2.56	274.42	8.32
8708	Boiler and blacksmith shop	17	23.41	1.38	44.50	2.62	67.91	4.00
8722	Machine shop	20	55.84	2.79	135.01	6.75	190.85	9.54
8812.50	Warehouse	26	45.09	1.73	70.48	2.71	115.57	4.45
8846	Sample room	7	22.97	3.28	52.83	7.54	75.80	10.82
9017.12	Feed water heating plant	6	53.35	8.89	25.83	4.31	79.18	13.20
9019	Power house	94	307.86	3.28	1,449.97	15.42	1,757.83	18.70
	Total	563	$2,726.18	$4.84	$3,292.57	$5.85	$6,018.75	$10.69

Cost and Erection of Machinery

Number	Name of Account	Cwt.	Labor Cost Amount	Labor Cost per Cwt.	Erection Cost Amount	Erection Cost per Cwt.	Total Cost Amount	Total Cost per Cwt.
	Group No. 1							
8313–8317	Two electrical feed pumps	433.45	$375.70	$0.87	$375.70	$0.87	$6,187.56	$14.28
8316	Six No. 14 Wilgus oil systems	84.75	111.48	1.32	123.82	1.46	1,973.77	23.29
8317.1	Two steam feed pumps	35.47	37.27	1.05	38.12	1.07	499.24	14.07
9006.1	Two Nordberg blowers with air receivers	3,832.42	1,641.62	0.43	3,080.52	0.80	34,155.64	8.91
9007.1	Three Curtis turbines and ten auto transformers	4,541.40	2,297.70	0.51	4,442.20	0.98	81,884.19	18.03
9009.12	Two dry vacuum pumps for jet condenser	242.00	285.51	1.18	517.38	2.14	3,145.52	13.00
9009.22	Two circulating pumps	375.60	366.90	0.98	433.78	1.15	3,902.58	10.39
9010.02	Air compressor	978.40	642.90	0.66	791.57	0.81	.....	.....
9011.03	Three dry vacuum pumps	140.00	147.26	1.05	196.78	1.42	3,337.36	23.84
9011.04	Three pumps and engines	972.55	389.32	0.40	568.75	0.58	9,118.69	9.38
9060.02	Two 5 by 8 vertical triplex pumps	113.54	176.20	1.55	195.07	1.72	2,211.78	19.48
		11,749.58	$6,471.86	$0.55	$10,763.69	$0.92	$146,416.33	$13.59

Group No. 2

Number	Name of Account	Cwt.	Labor Cost Amount	Labor Cost per Cwt.	Erection Cost Amount	Erection Cost per Cwt.	Total Cost Amount	Total Cost per Cwt.
8406	Two 40-ton Morgan cranes	2,215.00	$1,438.50	$0.65	$3,813.34	$1.72	$23,027.65	$10.40
8407	Two clinkering machines	1,692.13	1,715.23	1.01	2,435.73	1.44	15,697.17	9.28
8415	Two casting machines.	2,692.20	3,266.34	1.21	3,682.40	1.37	27,477.55	10.21
		6,599.33	$6,420.07	$0.97	$9,931.47	$1.50	$66,202.37	$10.03

Group No. 3

7704	Farrell crusher, 36 by 18	500.00	$392.86	$0.79	$401.31	$0.80	$1,486.47	$2.96
8112	Two motor-driven fans at roaster building	61.40	77.69	1.27	81.11	1.32	1,483.60	24.16
8718	Traveling hand crane, 5 ton	30.00	25.19	0.84	44.90	1.50	589.55	19.65
9004	" 20 ton	252.00	131.89	0.52	169.16	0.67	1,855.16	7.36
9008.1	Three surface condensers	1,157.00	415.31	0.36	542.82	0.47	19,978.86	17.27
9009.03	One barometric condenser	81.32	128.97	1.59	185.33	2.28	1,078.65	13.26
		2,081.72	$1,171.91	$0.56	$1,424.63	$0.68	$26,472.29	$12.72

Group No. 4

9011.02	Two exciters	543.00	$491.01	$0.90	$864.31	$1.59	$6,609.27	$12.17
9012.02	Two 150 Kw. synchronous generator sets	418.98	319.06	0.76	699.23	1.67	7,149.39	17.06
		961.98	$810.07	$0.84	$1,563.54	$1.63	$13,758.66	$14.30

In the above average costs an effort has been made for a logical grouping, yet it is somewhat arbitrary. Group 1 contains the erection of engine machinery. It was here necessary, in addition to handling heavy weights and placing on the foundation, to clean, adjust, and line up many mechanical parts. Group 2 is very similar to 1, but the machinery is not of the engine type and not so heavy in proportion to the labor required to put it in working order. Group 3 comprises machinery that required little other labor in the main than the lifting of heavy loads into place. Group 4 is somewhat similar to Group 3, but the labor is principally electrical. The above costs are reported as labor, erection, and total costs. The labor cost is self explanatory. The erection cost is the labor cost plus the needed small supplies, such as waste, oil, small tools, and the like. The total cost is also self-explanatory.

Masonry

Number	Name of Account	Cu. Yd.	Labor Cost Amount	Labor Cost per Cu. Yd.	Material Cost Amount	Material Cost per Cu. Yd.	Total Cost Amount	Total Cost per Cu. Yd.
7312.2	Retaining wall	21.9	$88.08	$4.03	$47.51	$2.17	$135.59	$6.19

Cost of Painting Concrete

Number	Name of Account	Sq. Yd.	Labor Cost Amount	Labor Cost per Sq. Yd.	Material Cost Amount	Material Cost per Sq. Yd.	Total Cost Amount	Total Cost per Sq. Yd.
9002.45	Power house basement floor.. Two coats of Toch cement filler	830	$81.45	$0.10	$48.81	$0.06	$130.26	$0.16
9002.7	Power house walls and foundations. One coat of Wadsworth Howland Bay State cement paint	2,459	195.84	0.08	301.61	0.12	497.45	0.20
9003.43	Power house, top of main floor slab. Two coats of Toch cement filler—one coat of Toch cement paint	1,134	95.56	0.08	199.32	0.18	294.88	0.26

Painting Corrugated Iron

Number	Name of Account	Sq. Yd.	Labor Cost Amount	Labor Cost per Sq. Yd.	Material Cost Amount	Material Cost per Sq. Yd.	Total Cost Amount	Total Cost per Sq. Yd.
8812.30	Warehouse roof, underside...	813.	$81.16	$0.10	$65.66	$0.08	$146.82	$0.18
	Two coats white lead and oil							

Painting Berger Plate

9003.42	Power house main floor slab, underside.....	2,679	$181.88	$0.07	$147.58	$0.06	$329.46	$0.12
.54	Power house roof, underside..	6,813	692.84	0.10	324.55	0.05	1,017.39	0.15
	Two coats white lead and oil							
	Square yards derived by developing the Berger plate. This is three times the superficial area							

Miscellaneous Painting

Number	Name of Account	Sq. Yd.	Labor Cost Amount	Labor Cost per Sq. Yd.	Material Cost Amount	Material Cost per Sq. Yd.	Total Cost Amount	Total Cost per Sq. Yd.
8703.60	Boiler shop.....	1,574	$92.53	$0.05	$60.73	$0.04	$153.26	$0.10
8718.60	Machine shop.....	1,989	118.00	0.06	87.40	0.04	205.40	0.10
8813.10	Warehouse.....	412	26.50	0.06	14.17	0.03	40.67	0.10
		3,975	$237.03	$0.06	$162.30	$0.04	$399.33	$0.10
	This was two-coat work on wood							

Painting Window Sash and Doors

Number	Name of Account	No. of Sash	Labor Cost Amount	Labor Cost per Sash	Material Cost Amount	Material Cost per Sash	Total Cost Amount	Total Cost per Sash
7803.11	Sampling plant.....	129	$118.94	$0.92	$28.96	$0.22	$147.90	$1.14
8813.11	Warehouse.....	189	122.78	0.65	15.34	0.08	138.12	0.73
	Window sash and frames, two-coat work							
9003.60	Power house.....	299	290.09	0.97	16.72	0.05	306.81	1.02
	Window sash only, three-coat work							
	Doors figured as two sash							

Roofs

Number	Name of Account	Squares	Labor Cost Amount	Labor Cost per Sq.	Material Cost Amount	Material Cost per Sq.	Total Cost Amount	Total Cost per Sq.
8703.30	Boiler and blacksmith shop.....	66.49	$286.52	$4.31	$828.24	$12.46	$1,114.76	$16.77
8717.30	Machine and carpenter shop.....	77.21	297.85	3.86	953.04	12.34	1,250.89	16.20
		143.70	$584.37	$4.06	$1,781.28	$12.40	$2,365.65	$16.46
	Consists of 2 by 8 tongue and groove sheathing, asbestos roofing and nailing strips. No painting included							
8843.4	Sample room.....	8.0	$23.00	$2.88	$76.57	$9.57	$99.57	$12.45
	Consists of 1 by 12 sheathing and asbestos roofing. No painting included							
9003.5	Power house.....	214.83	$3,587.15	$16.70	$5,791.05	$26.96	$9,378.20	$43.65
	Consists of Berger plate, concrete, tar, and P and B roofing. Painting the underside of the Berger plate is included							
9017.04	Feed water heating plant	8.80	$115.25	$13.10	109.88	$12.48	$225.13	$25.58
9017.25	Condensed water pump house.....	5.00	73.92	14.78	60.83	12.17	134.75	26.95
		13.80	$189.17	$13.71	$170.71	$12.37	$359.88	$26.08
	Consists of 2 by 8 sheathing with asbestos roofing							
9060.07	Oil pump house.....	6.50	$136.81	$21.05	$162.66	$25.02	$299.47	$46.07
	Consists of "hyrib," concrete and plaster							
9060.131	Oil storage tanks.....	65.56	$761.95	$11.62	$882.84	$13.47	$1,644.79	$25.09
	Consists of wood supporting structure, 1-in. sheathing and metal lath and plaster							

Shafting, Pulleys and Belting

Number	Name of Account	Lin. Ft. of Shafting	Labor Cost Amount	Labor Cost per Ft.	Material Cost Amount	Material Cost per Ft.	Total Cost Amount	Total Cost per Ft.
7804	Sampling plant	85	$64.01	$0.75	$1,871.07	$22.01	$1,935.08	$22.76
8107	Roasting plant	164	118.24	0.72	1,999.89	12.20	2,118.13	12.92
8706	Boiler and blacksmith shop	51	105.59	2.07	301.16	5.91	406.75	7.98
8720	Machine and carpenter shop	162	289.29	1.79	1,513.36	9.34	1,802.65	11.13
8849.2	Sample room	26	23.39	0.89	170.17	6.55	193.56	7.44

Structural Steel

Number	Name of Account	Amount	Tons	Cost per Ton
7308 2	Trestle approach to reverb. building	$13,460.84	163.97	$82.09
7310	Bridge No. 1	377.40	3.70	102.00
7314	Trestles to receiving bins	9,269.48	109.35	84.77
7403	Receiving bins	29,276.63	353.09	82.92
7703	Crushing plant	2,420.36	25.07	96.54
7803	Sampling plant	10,408.12	110.85	93.89
7903	Bedding plant	47,404.86	548.71	86.39
8103	Roasting plant	37,252.67	445.28	83.66
8123	Roaster dust chamber	34,745.41	415.68	83.59
8303	Reverberatory plant	40,799.76	461.09	88.48
8306.2	Flues from boilers to reverb. flue	2,815.32	34.78	80.95
8308	Boiler building	25,839.85	292.03	88.48
8403	Converter plant	69,359.60	783.86	88.48
8423	Converter dust chamber	20,371.20	238.30	85.49
8503	Conveying system	19,365.98	211.73	91.47
8614	Reverberatory flue	3,593.06	41.61	86.35
8624	Converter flue	6,616.44	81.99	80.70
8629	Roaster dust chamber flue	9,209.36	94.46	97.49
8703	Boiler and blacksmith shop	2,913.90	32.72	89.06
8717	Machine and carpenter shop	3,431.42	38.23	89.76
8812	Warehouse	3,734.08	39.76	93.92
9003	Power house	23,773.13	254.29	93.49
9014.02	North and south mains	7,694.58	86.81	88.64
9017.02	Feed water heating plant	2,262.90	26.63	84.98
	Total	$426,396.35	$4,839.99	$87.13

Tile Walls

Number	Name of Account	Cu. Ft.	Labor Cost Amount	Labor Cost per Cu. Ft.	Material Cost Amount	Material Cost per Cu. Ft.	Total Cost Amount	Total Cost per Cu. Ft.
8106.02	Roaster flues	2,365	$374.82	$0.16	$400.36	$0.17	$775.18	$0.33
8123.10	Roaster dust chamber.	14,980	2,268.29	0.15	2,585.45	0.17	4,853.74	0.32
8423.10	Converter dust chamber	6,369	1,620.05	0.25	2,182.40	0.34	3,802.45	0.60
8613	Reverberatory flue	6,400	1,272.86	0.20	1,696.39	0.26	2,969.25	0.46
8628	Roaster dust chamber flue	4,231	1,018.13	0.24	1,573.06	0.37	2,591.19	0.61
8703.20	Boiler and blacksmith shop	2,297	477.95	0.21	612.62	0.27	1,090.57	0.48
8717.20	Machine and carpenter shop	2,397	531.45	0.22	571.28	0.24	1,102.73	0.46
8812.20	Warehouse	2,342	438.00	0.19	477.86	0.20	915.86	0.39
9003.10	Power house	14,343	3,856.83	0.26	4,510.20	0.31	8,367.03	0.58
9017.03	Feed water heating plant	706	285.99	0.41	234.36	0.33	520.35	0.74
9017.242	Condensed water pump house	257	98.87	0.38	84.28	0.33	183.15	0.71
	Total	56,687	$12,243.24	$0.22	$14,928.26	$0.26	$27,171.50	$0.48

Unloading Brick and Tile

Number	Name of Account	Total Cost	Tons	Cost per Ton
8105.01	Roaster brick	$363.14	1,231.01	$0.21
8123.11	Roaster dust chamber tile	307.72	525.05	0.58
8304.01	Reverberatory furnaces	1,264.49	2,279.49	0.55
8305.01	Cross and header flue	497.60	1,329.23	0.37
8309.02	Waste heat boilers	518.91	1,073.74	0.48
8309.12	Oil-fired boilers	183.94	228.11	0.81
8405.11	Converters	186.25	579.30	0.32
8423.11	Converter dust chamber	37.61	155.20	0.24
8613.01	Reverberatory flue	57.79	278.33	0.21
8628.01	Roaster dust chamber flue	57.60	171.40	0.34
8703.21	Boiler and blacksmith shop	18.89	69.70	0.27
8717.21	Machine and carpenter shop	44.06	58.80	0.75
8812.21	Warehouse	16.50	74.20	0.22
9003.11	Power plant	332.57	522.70	0.64
9017.031	Feed water heating plant	24.91	30.00	0.83
	Total	$3,911.98	8,606.26	$0.45

These costs cover such costs as cleaning the site for unloading, building runways where needed, constructing brick sheds, and checking the quantities of the shipment, as well as the unloading of the various shapes in separate piles.

Ventilators

Number	Name of Account	No. of Ventilators	Labor Cost Amount	Labor Cost per Vent	Material Cost Amount	Material Cost per Vent	Total Cost Amount	Total Cost per Vent
8703.31	Boiler and blacksmith shop	3	$16.01	$5.33	$261.50	$87.17	$277.51	$92.50
8717.31	Machine and carpenter shop	3	11.16	3.72	248.24	82.74	259.40	86.46
		6	$27.17	$4.53	$509.74	$84.96	$536.91	$89.49
	48-in. Burt ventilators, square base, set on wooden roof							
8812.31	Warehouse	3	$30.38	$10.13	$207.12	$69.04	$237.50	$79.17
	48-in. Burt ventilators, round base, set on corrugated iron roof							
9003.3	Power house	6	$125.60	$20.94	$439.76	$73.29	$565.36	$94.23
	48-in. Burt ventilators, square base, set on concrete roof							
9017.045	Feed water heating plant	2	$142.14	$71.07	$49.20	$24.60	$191.34	$95.67
	42-in. ventilators made in new smelter shops, set on wooden roof							

Windows and Doors

Number	Name of Account	Sq. Ft. Opening	Labor Cost Amount	Labor Cost per Sq. Ft.	Material Cost Amount	Material Cost per Sq. Ft.	Total Cost Amount	Total Cost per Sq. Ft.
7703.1	Crushing plant	529	$84.00	$0.16	$170.71	$0.32	$254.71	$0.48
7803.1	Sampling plant	2,086	332.94	0.16	564.90	0.27	897.84	0.43
		2,615	$416.94	$0.16	$735.61	$0 28	$1,152.55	$0.44
	The above accounts cover wooden sash and frame set in steel and corrugated iron building							
8843.5	Sample room	298	$32.39	$0.11	$118.85	$0.40	$151.24	$0.51
	The above account covers wooden sash and frame set in wood and corrugated iron building							
8703.1	Boiler and blacksmith shop	2,581	$693.02	$0.27	$2,456.28	$0.95	$3,149.30	$1.22
8717.1	Machine and carpenter shop	3,037	923.61	0.30	2,992.16	0.99	3,915.77	1.29
		5,618	$1,616.63	$0.29	$5,448.44	0.97	$7,065.07	$1.26
	The above accounts cover steel sash and frame set in steel and tile curtain wall building							

Windows and Doors (*Continued*)

Number	Name of Account	Sq. Ft. Opening	Labor Cost Amount	Labor Cost Per Sq. Ft.	Material Cost Amount	Material Cost Per Sq. Ft.	Total Cost Amount	Total Cost Per Sq. Ft.
8812.1	Warehouse	1,982	$533.02	$0.27	$1,056.31	$0.53	$1,589.33	$0.80
8822.5	Laboratory	823	208.70	0.25	480.33	0.59	689.03	0.84
9017.035	Heater house	186	59.83	0.32	99.98	0.54	159.81	0.86
9017.24	Cond. water pump house.	57	26.02	0.45	22.68	0.40	48.70	0.85
		3,048	$827.57	$0.27	$1,659.30	$0.54	$2,486.87	$0.81
	The above account covers wooden sash and frame set in steel and tile curtain wall building							
9060.06	Oil pump house	124	$83.60	$0.68	$65.02	$0.52	$148.62	$1.20
	The above account covers wooden sash and frame set in concrete walls							
9003.2	Power house	4,044	$974.38	$0.24	$3,319.93	$0.82	$4,294.31	$1.06
	The above account covers steel frame and wooden sash set in steel and tile curtain wall building							

Wire Baffles

Number	Name of Account	C. Wires	Labor Cost Amount	Labor Cost Per C. Wires	Material Cost Amount	Material Cost Per C. Wires	Total Cost Amount	Total Cost Per C. Wires
8123.01	Roaster dust chamber..	604.80	$523.63	$0.87	$4,758.23	$7.87	$5,281.86	$8.73
8423.01	Converter dust chamber	166.10	138.86	0.84	1,101.95	6.63	1,240.81	7.47
		770.90	$662.49	$0.86	$5,860.18	$7.60	$6,522.67	$8.46

Woodwork

Number	Name of Account	M.B.M.	Labor Cost Amount	Labor Cost Per M.B.M.	Material Cost Amount	Material Cost Per M.B.M.	Total Cost Amount	Total Cost Per M.B.M.
7308.3	Trestle approach to reverb. bldg	27.65	$703.93	$25.45	$768.92	$27.81	$1,472.85	$53.27
7314.3	Trestle to receiving bins	27.21	572.23	21.03	838.11	30.80	1,410.34	51.83
		54.86	$1,276.16	$23.26	$1,607.03	$29.29	$2,883.19	$52.55
9052	Cooling tower	128 63	3,181.33	24.73	5,415.47	42.10	8,596.80	66.84
9060.16	Track bumpers	3.16	246.74	78.08	47.92	15.16	294.66	93.25

Wooden Floors

Number	Name of Account	Sq. Ft.	Labor Cost Amount	Labor Cost Per Sq. Ft.	Material Cost Amount	Material Cost Per Sq. Ft.	Total Cost Amount	Total Cost Per Sq. Ft.
8717.40	Machine and carpenter shop. See sketch No. 24	4,136	$269.80	$0.07	$593.30	$0.14	$863.10	$0.21

CHAPTER III

COMPOSITE COSTS

Cost of Buildings per Square Foot of Floor Space

Number	Name of Account	Sq. Ft. of Floor Space	Total Cost Amount	Total Cost Per Sq. Ft.
7700	Crushing plant	1,650	$5,968.32	$3.62
	Includes accounts 7701 to 7703.2 inclusive and 7707.1			
7800	Sampling plant	6,140	16,299.16	2.65
	Includes accounts 7801 to 7803.11 inclusive, 7806.1 and 7809			
8100	Roasting plant	28,740	43,322.75	1.51
	Includes accounts 8101 to 8103.1 inclusive, and 8109			
8300	Reverberatory plant	20,370	50,687.28	2.49
	Includes accounts 8301 ($784.32), 8302 ($8,560.45), 8303 and one-half of 8313.1			
8307	Reverberatory boiler building	14,310	36,887.67	2.58
	Includes accounts 8307; 0.04; 0.1; 0.2; 0.3; 8308, one-half of 8313.1			
8400	Converter building	26,084	87,231.14	3.34
	Includes accounts 8401, 8402, 8403, 8409.1, 8413, 8414, 8416, 8416.1, 8416.11 and 8416.2			
8700	Boiler and blacksmith shop	4,424	11,320.58	2.56
	Includes accounts 8701 to 8703.40 inclusive, 8703.60 and 8708			
8714	Machine and carpenter shop	5,144	14,905.56	2.90
	Includes accounts 8714 to 8717.50 inclusive; 8717.60 and 8722			
8809	Warehouse	5,040	11,512.93	2.28
	Includes accounts 8810 to 8812.50 inclusive; 8813.10 and 8813.11			
8819	Laboratory	1,492	11,363.77	2.92
	Includes accounts 8820 to 8822.5 inclusive; 8825, 8828 and 8829			
8840	Sample room	600	991.46	1.65
	Includes accounts 8841 to 8843.5 inclusive; 8846 and 8848			
9000	Power plant	32,096	77,452.56	2.41
	Includes accounts 9001 to 9003.61 inclusive, excluding 9002.2			

Cost of Buildings per Cubic Foot

Number	Name of Account	Cu. Ft. in Building	Total Cost Amount	Total Cost Per Cu. Ft.
7700	Crushing plant	27,040	$5,968.32	$0.22
7800	Sampling plant	80,547	16,299.16	0.20
8100	Roasting plant	410,140	43,322.75	0.11
8300	Reverberatory plant	474,350	50,687.28	0.11
8307	Reverberatory boiler building	500,850	36,887.67	0.07
8400	Converter building	1,529,636	87,231.14	0.06
8700	Boiler and blacksmith shop	86,268	11,320.58	0.15
8714	Machine and carpenter shop	100,308	14,905.56	0.15
8809	Warehouse	83,160	11,512.93	0.14
8819	Laboratory	16,140	4,363.77	0.27
8840	Sample room	6,000	991.46	0.16
9000	Power house	784,000	77,452.56	0.10

In the above costs the same account numbers are used as in computing the cost of buildings per square foot of floor space.

Cost of Buildings Equipped per Square Foot of Floor Space

Number	Name of Account	Sq. Ft. of Floor Space	Total Cost Amount	Total Cost Per Sq. Ft.,
7700	Crushing plant	1,650	$9,268.62	$5.62
	Accounts 7701 to 7707.1 inclusive			
7800	Sampling plant	6,140	34,108.74	5.56
	Accounts 7801 to 7810 inclusive			
8100	Roasting plant	28,740	136,734.87	4.76
	Accounts 8101 to 8113.2 inclusive			
8300	Reverberatory plant	20,370	172,171.55	8.45
	Accounts 8301 to 8305.2 inc.—8307.2, 8312 and 8313, deducting one-half of each—8314, 8315 and 8318—8316, deducting two-thirds of this account			
8307	Reverberatory boiler building	14,310	159,716.26	11.16
	Accounts 8306 to 8317.2 inc. deducting one-half of 8307.2, 8312 and 8313.1, and two-thirds of 8316			
8400	Converter building	26,084	216,033.37	8.28
	Accounts 8401 to 8419.4 and 8425 to 8426.4 inclusive			
8700	Boiler shop	4,424	21,449.23	4.85
	Accounts 8701 to 8708 inclusive			
8714	Machine and carpenter shop	5,144	27,356.27	5.32
	Accounts 8715 to 8722 inclusive			
8809	Warehouse	5,040	13,602.71	2.70
	Accounts 8810 to 8813.11 inclusive			
8819	Laboratory	1,492	6,144.02	4.12
	Accounts 8820 to 8831 inclusive			
8840	Sample room	600	2,826.11	4.71
	Accounts 8841 to 8849.2 inclusive			
9000	Power house	32,096	359,590.10	11.20
	Accounts 9001 to 9004 and 9006.01 to 9016.01 inclusive, deducting one-half of 9014 to 9014.05 inclusive.			

Cost of Buildings Equipped per Cubic Foot.

Number	Name of Account	Cu. Ft. of Floor Space	Total Cost Amount	Total Cost Per Cu. Ft.
7700	Crushing plant	27,040	$9,268.62	$0.34
7800	Sampling plant	80,547	34,108.74	0.42
8100	Roasting plant	410,140	136,734.87	0.33
8300	Reverberatory plant	474,350	172,171.55	0.36
8307	Reverberatory boiler building	500,850	159,716.26	0.32
8400	Converter building	1,529,636	216,033.37	0.14
8700	Boiler and blacksmith shop	86,286	21,449.23	0.24
8714	Machine and carpenter shop	100,308	27,356.27	0.27
8809	Warehouse	83,160	13,602.71	0.16
8819	Laboratory	16,140	6,144.02	0.38
8840	Sample room	6,000	2,826.11	0.47
9000	Power house	784,000	359,590.10	0.46
	The amounts shown above are the same as those used in the cost of buildings equipped per square foot			

Bedding Plant and Bunker Bins. Cost per Cubic Foot of Capacity of both Beds and Bins

Capacity	Total Cost Amount	Total Cost Per Cu. Ft.
228,440 cu. ft.	$150,939.05	$0.66
This cost consists of accounts 7901 to 7908 inclusive		
The capacity was obtained by actual measurement		

Three Spreading Beds. Cost per Cubic Foot of Capacity

Capacity	Total Cost Amount	Total Cost Per Cu. Ft.
160,380 cu. ft.	$120,177.94	$0.75
This is partly an estimated cost. It was obtained by deducting the cost of the bunker bins installation from the total bedding plant, and substituting therefor such excavation, foundation, steel work, etc., as would be necessary to provide for conveyors 7[1] and 10[1]		

Receiving Bins—Cost per Cubic Foot of Capacity

Capacity	Total Cost Amount	Per Cu. Ft
11,386 cu. ft.	$38,073.73	$3.34

This cost consists of accounts 7401 to 7404 inclusive, and 7407

The capacity of the receiving bins was obtained by assuming that the material lay on a one and one-half to one slope on falling from the cars

Conveyors—Cost per Ton of Capacity

Conveyors	Capacity, Tons per Hour	Total Cost Amount	Per Ton	
No. 1	100	$3,258.11	$32.58	Account 7405
" 2	150	2,853.22	19.02	" 7405.01
" 7^1–7^2	250	6,938.01	27.75	" 7904
" 8^1–8^2–8^3	450	9,929.99	22.07	" 7904.1
" 9^1–9^2–9^3–10^1–10^2	500	11,668.39	23.33	" 7904.2
" 12	100	969.20	9.69	" 8113
" 13^1–13^2	200	3,948.62	19.74	" 8113.1
" 15	100	2,674.24	26.74	" 8425
" 3–4–5–6–11–14	700	13,716.49	19.59	" 8505

The above costs do not include any steel supporting structure for the conveyors. Capacities are taken from the designers' drawings

Complete Conveyor—Cost per Lineal Foot

Conveyor No.	Total Length	Amount	Cost per Foot
No. 3, 4, 5, 6, 11, 14	1,284.9 ft.	$44,290.65	$34.47

Cost consists of accounts 8501 to 8505.2, and 8506. The total length is the sum of the lengths of the various conveyors measured from center of head pulley to center of tail pulley

Cost of Cooling Tower per Thousand Gallons a Minute

M Gallons per Min.	Total Cost Amount	Per M Gallons
12 M	$26,273.01	$2,189.42

Cost consists of accounts 9050 to 9053 inclusive

Cost of Dust Chambers per Cubic Foot

Name	Cu. Ft.	Total Cost Amount	Per Cu. Ft.	
Roaster dust chamber	256,860	$49,664.76	$0.19	Accounts 8121 to 8123.3 inclusive
Converter dust chamber	67,210	27,813.58	0.41	" 8421 to 8428 "

Cost of Flues per Cubic Foot

Name	Cu. Ft.	Total Cost Amount	Per Cu. Ft.	
Reverberatory flue	63,420	$13,453.70	$0.21	Accounts 8611 to 8614.2 inclusive
Roaster dust chamber flue	29,527	12,859.10	0.44	" 8625 to 8629 "
Flues from boilers to reverb. flues	6,734	2,983.42	0.44	" 8306 to 8306.2 "
Converter flue	10,705	7,602.88	0.71	" 8621 to 8624 "

Cost of Flues per Lineal Foot

Name	Lin. Ft.	Total Cost Amount	Per Ft.	
Reverb. flue	250	$13,453.70	$53.81	In obtaining these costs the same
Roaster dust chamber flue	170	12,859.10	75.64	accounts were used as in com-
Flues from boilers to reverb.	246	2,983.42	12.13	puting the cost of flue per cubic
Converter flue	208	7,602.88	36.55	foot.

Cost of Power Plant per Indicated Horse Power, Boiler Plant Included

I.H.P.	Total Cost Amount	Per I.H.P.
10,660	$589,717.16	$55.32

Cost consists of accounts 8306 to 8313.1 inclusive—deducting one-half of 8307.2, 8312 and 8313.1. 8316 to 8317.2 inclusive—deducting two-thirds of 8316. 9001 to 9004 inclusive. 9006.01 to 9053 inclusive. 9060.20 to 9060.23 inclusive—deducting three-fourths of these accounts.

	I.H.P.
3 turbines	9,460
2 Nordberg blowers	1,000
1 air compressor	200
Total	10,600

Cost of Power Plant per Indicated Horse Power, Boiler Plant not Included

I.H.P.	Total Cost Amount	Per I.H.P.
10,660	$398,631.17	$37.40

Cost consists of accounts 9001 to 9004 inclusive. 9006.01 to 9016.02 inclusive—deducting one-half of 9014 to 9014.05. 9017.20 to 9017.27 inclusive. 9018.1 to 9022 inclusive. 9050 to 9053 inclusive

Cost of Boiler Plant per Boiler Horse Power

Boiler Horse Power	Total Cost Amount	Per Horse Power
6,143	$191,085.99	$31.11

Cost consists of accounts 8306 to 8313.1 inclusive—deducting one-half of 8307.2, 8312 and 8313.1. 8316 to 8317.2 inclusive—deducting two-thirds of 8316. 9014 to 9014.05 inclusive—deducting one-half of these accounts. 9017 to 9017.15 inclusive. 9060.20 to 9060.23—deducting three-quarters of these accounts.

Boiler Horse Power

7 waste heat @ 713	4,991
3 oil-fired @ 384	1,152
Total boiler horse power	6,143

Cost of Reverberatory Plant per Ton of Capacity

Tons per 24 Hr.	Total Cost Amount	Per Ton
1,200	$172,171.55	$143.47

Cost includes accounts 8301 to 8305.2 inclusive. 8307.2, 8312, 8313.1—(one-half of each). 8314, 8315, and 8318. 8316—(two-thirds). 3 reverberatories at 400 tons per 24 hr. capacity.

Cost of Complete Roasting Installation per Roaster

No. of Roasters	Total Cost Amount	Per Roaster
8	$136,734.87	$17,091.86

Cost consists of accounts 8101 to 8113.2—Roaster building and roasters

8	$199,258.73	$24,907.34	3

Cost consists of accounts 8101 to 8113.2 inclusive—Roaster building and roasters. 8121 to 8123. inclusive—Roaster dust chamber. 8626 to 8629 inclusive—Roaster dust chamber flue.

Tracks—Cost per Foot

	Labor Cost		Material Cost		Total Cost	
Length of Track	Amount	Per Foot	Amount	Per Foot	Amount	Per Foot
14,116 ft.	$38,190.18	$2.71	$27,335.46	$1.94	$65,525.64	$4.64

This cost consists of accounts 7301 to 7305 inclusive. The 697 ft. of track which was on trestles was deducted from account 7303.

CHAPTER IV

WAGE SCALE

Occupation	Feb. 28, 1912, Ten Hours		April 1, 1912, Nine Hours		July 24, 1912, Nine Hours		July 1, 1913, Nine Hours		Sept. 1, 1913, Eight Hours	
	A.	M.	A.	M.	A.	M.	A.	M.	A.	M.
Blacksmiths	.....	$2.50	$4.00	.....	$4.50	.....	$4.50	.....	$4.25	.....
" "	.....	.....	.....	.....	.....	.....	.....	.....	4.50	.....
" helpers	.....	.....	.....	2.25	3.00	3.00	2.50	2.50	.....	.....
" "	.....	.....	.....	.....	.....	.....	3.00	3.00	2.50	2.50
" "	.....	.....	.....	.....	.....	.....	3.50	.....	3.25	3.25
Boilermaker boss	.....	.....	.....	.....	.....	.....	5.50	.....	5.50	.....
Boilermakers	.....	.....	.....	.....	4.50	.....	4.50	.....	4.25	.....
" " layer-out	.....	.....	.....	.....	.....	.....	4.75	.....	4.75	.....
" " helpers	.....	.....	.....	.....	3.00	.....	3.00	3.00	3.00	3.00
Brick masons	.....	.....	.....	.....	.....	.....	[1]6.50	.....	[1]6.50	.....
" tenders	.....	.....	.....	.....	.....	.....	.....	[1]2.25	.....	[1]2.25
Carpenter boss	.....	.....	.....	.....	5.00	.....	5.00	.....	5.00	.....
" 1st class	4.50	.....	4.00	.....	4.50	.....	4.50	.....	4.25	.....
" 2nd class	4.00	4.00	.....	.....	.....	.....	.....	.....	.....	.....
" helpers	3.00	3.00	3.00	3.00	3.00	.....	3.00	.....	2.50	.....
" (with tools)	.....	.....	.....	.....	3.50	.....	3.50	.....	3.00	.....
" helpers	.....	.....	.....	2.25	.....	.....	.....	.....	.....	.....
Cart drivers	.....	.....	.....	.....	2.25	2.25	.....	.....	.....	.....
Cement finisher boss	.....	.....	.....	.....	.....	.....	.....	.....	5.00	.....
" "	.....	.....	.....	.....	4.50	.....	.....	2.50	.....	.....
" "	.....	.....	.....	.....	.....	.....	3.50	3.50	.....	.....
" "	.....	.....	.....	.....	.....	.....	4.00	4.00	.....	.....
Concrete boss	.....	.....	.....	.....	4.50	.....	4.50	.....	.....	.....
" mixers	.....	.....	.....	2.25	.....	2.25	.....	2.25	.....	.....
Corral boss	.....	.....	3.50	.....	3.50	.....	3.50	.....	3.50	.....
" men	.....	.....	.....	.....	2.25	2.25	.....	.....	.....	.....
Drillers	.....	2.25	.....	2.00	.....	.....	.....	.....	.....	.....
Electrician boss	.....	.....	.....	.....	.....	.....	5.00	.....	4.50	.....
" "	.....	.....	.....	.....	.....	.....	.....	.....	5.00	.....
Electrician	.....	.....	.....	.....	.....	.....	4.50	.....	4.25	.....
" helpers	.....	.....	.....	.....	.....	.....	3.00	2.25	2.75	.....
" "	.....	.....	.....	.....	.....	.....	.....	.....	3.00	.....
Engineers, locomotive	3.25	.....	.....	.....	.....	.....	.....	.....	.....	.....
" compressors	.....	.....	.....	.....	3.50	.....	3.50	.....	3.50	.....
" stationary	.....	.....	.....	.....	3.00	.....	.....	.....	.....	.....
Janitors	.....	.....	.....	2.00	.....	2.25	.....	.....	.....	.....
Labor bosses	.....	.....	3.25	2.00	4.00	.....	4.00	.....	4.00	.....
" "	.....	.....	4.00	2.50	.....	.....	4.50	.....	.....	.....
" "	.....	.....	.....	3.00	.....	.....	.....	.....	.....	.....
Laborers	.....	1.75	.....	1.75	.....	2.00	.....	2.00	.....	1.75
"	.....	.....	.....	.....	.....	.....	.....	.....	.....	2.00
Machine shop boss	.....	.....	.....	.....	.....	.....	5.00	.....	5.00	.....
" " " (outside)	.....	.....	.....	.....	.....	.....	5.50	.....	.....	.....
Machinist	.....	.....	4.00	.....	4.50	.....	4.50	.....	4.25	.....
" helpers	.....	.....	3.00	.....	3.00	3.00	3.00	2.50	3.00	3.00
Miners, underground	.....	.....	.....	[1]2.75	.....	2.75	.....	.....	.....	.....
" surface	.....	.....	.....	2.25	.....	2.25	.....	.....	.....	.....
Office boys	.....	.....	.....	.....	.....	.....	.....	2.00	.....	[2]2.00
" "	.....	.....	.....	.....	.....	.....	.....	.....	.....	2.25
Pipe fitter boss	.....	.....	.....	.....	.....	.....	5.50	.....	5.50	.....
Pipe fitter	.....	2.50	.....	.....	.....	.....	4.50	.....	4.25	.....
" " helpers	.....	.....	.....	.....	.....	.....	3.00	2.25	3.00	.....

A—American. M—Mexican. [1] Eight hours. [2] Nine hours.

Wage Scale (*Continued*)

Occupation	Feb. 28, 1912, Ten Hours		April 1, 1912, Nine Hours		July 24, 1912, Nine Hours		July 1, 1913, Nine Hours		Sept. 1, 1913, Eight Hours	
Plow holders	.....	.....	2.25	2.25	.....	.....	.....	.....	.....	.....
Plumbers	.....	.....	.....	.....	4.50	.....	.....	.....	.....	.....
" helpers	.....	.....	.....	.....	3.00	.....	.....	.....	.....	.....
Rigger boss	.....	.....	4.50	.....	5.00	.....	.....	.....	.....	.....
Rigger	.....	.....	.....	.....	.....	.....	3.50	.....	4.25	.....
"	.....	.....	.....	.....	.....	.....	4.50	.....	.....	.....
" helpers	.....	.....	2.25	2.25	.....	.....	2.25	2.25	2.25	2.25
" "	.....	.....	.....	.....	.....	.....	2.50	2.50	3.50	3.50
" "	.....	.....	.....	.....	.....	.....	3.00	3.00	.....	.....
Stone mason boss	.....	.....	.....	3.50	.....	.....	.....	.....	.....	.....
" "	.....	3.00	.....	3.00	.....	3.00	.....	.....	.....	.....
Steam fitters	.....	.....	.....	.....	4.50	.....	.....	.....	.....	.....
" helpers	.....	.....	.....	.....	3.00	.....	.....	.....	.....	.....
Teamsters, 4 and 6 head	.....	.....	3.00	.....	3.00	.....	.....	.....	.....	.....
" 2 head	.....	.....	2.00	2.00	2.25	2.25	3.00	.....	3.00	.....
" fresnos and slips	2.25	2.25	2.00	2.00	2.50	2.50	.....	.....	.....	.....
" plow	2.75	.....	2.25	2.25	.....	.....	.....	.....	.....	.....
Tinners	.....	.....	.....	.....	4.00	.....	4.50	.....	4.25	.....
" helpers	.....	.....	.....	.....	2.50	2.50	.....	.....	3.00	.....
Tool room man	.....	.....	.....	2.00	2.25	.....	.....	.....	.....	.....
Tool sharpeners	.....	.....	.....	2.50	3.50	.....	4.00	.....	.....	.....
Track boss	.....	.....	.....	.....	4.25	.....	.....	.....	.....	.....
Water boys	.....	1.30	.....	1.00	2.00	2.00	2.00	2.00	1.75	1.75
Warehouse help	.....	.....	.....	.....	.....	.....	3.00	2.50	3.00	.....

CHAPTER V

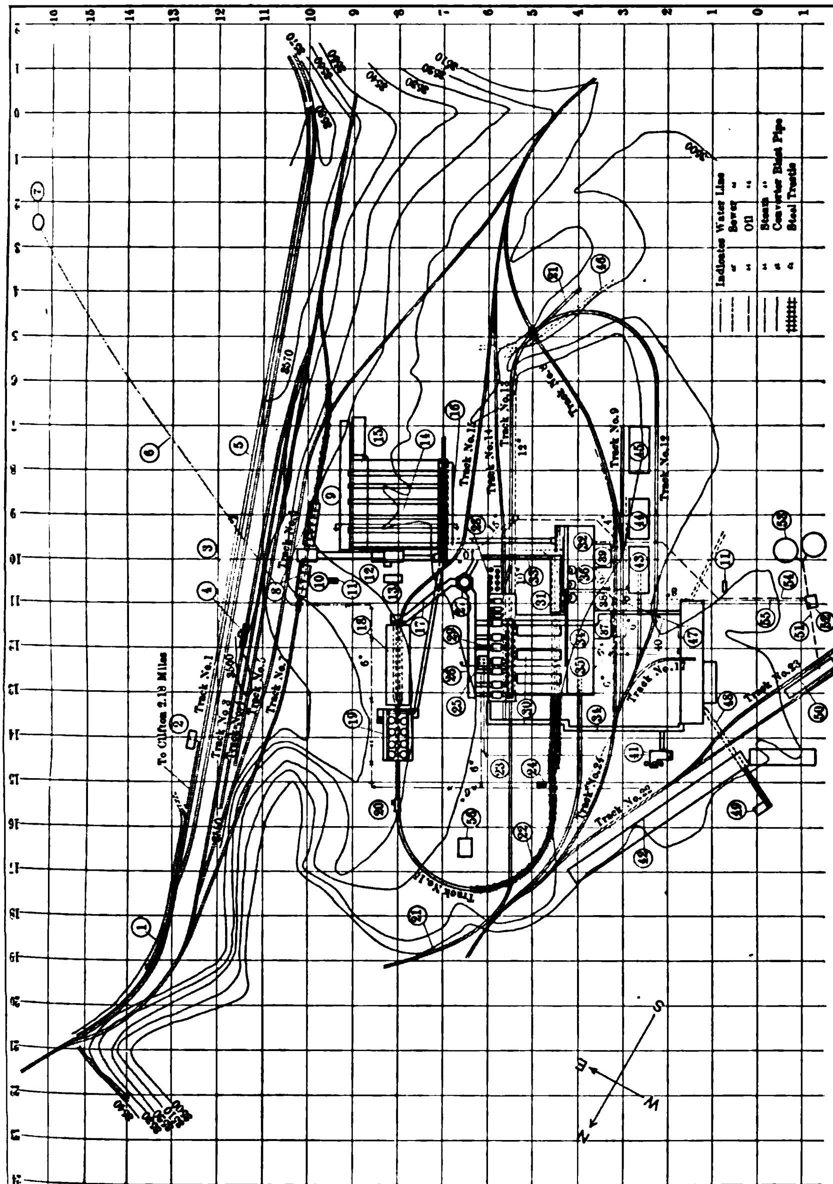
RAW MATERIAL PRICES

Prices of Raw Material F. O. B. Clifton During New Smelter Construction

Name	Price	Unit
Asphalt (Trinidad)	$3.62	Cwt.
Asbestos sheets	5.68	"
Brick, red standard, 2" × 4" × 8"	16.986	M.
" silica A1 straights, 2½" × 6" × 12"	98.10	"
" " B 2 " 2½" × 4½" × 9"	52.20	"
" " C 2 " 3" × 6" × 20"	184.19	"
" " D 1 " 3" × 6" × 15"	148.62	"
" Fire "Star," 2½" × 4½" × 9"	49.68	"
" " "Athens," 2½" × 4½" × 9"	40.00	"
" Magnesite straights, 2½" × 4½" × 9"	174.40	"
" " Special shapes at rate of $189.40 per M brick 2½" × 4½" × 9", figuring from cubical contents, *e.g.* "A" special is 2.3796 larger than a 2½" × 4" × 9" brick, therefore would cost 2.3796 × $189.40.		
Bolts, carriage ½" × 2"	0.97	C.
" machine ½" × 2"	1.01	"
" " ¾" × 2"	2.20	"
" " ½" × 24	13.04	"
Belting, conveyor rubber, 7–5–$\frac{4}{16}$", 30" wide	3.63	Lin. ft.
" " " 5–3–$\frac{4}{16}$", 20" "	1.9168	"
" " " 6–4–$\frac{4}{16}$", 20" "	2.1898	"
Castings, rough iron	2.00 to 3.00	Cwt.
Cement, "El Toro" brand	0.7225	sack

Prices of Raw Material F. O. B. Clifton During New Smelter Construction (*Con.*)

Name	Price	Unit
Clay, fire	$7.00	ton
Coal, steam	8.00 to 7.175	"
Conduit, ½" galvanised	6.39	C. ft.
Gasoline	0.23	gal.
Grease, "Arctic" No. 4	6.50	cwt.
Glass, factory ribbed, 13½" × 20"	13.28	C.
Hay, alfalfa	20.35	ton
Iron, corrugated No. 22, 10-ft. lengths	0.9729	sheet
" galvanised No. 20	3.49	cwt.
" round and square, base	2.25	"
" flat, base	2.35	"
" sheets	2.35	"
Lumber, common R. O. P.	26.00	M.
" No. 2 T & G flooring	31.76	"
Lead, white	8.80	cwt.
Lime, burnt	10.00	ton
Nails, common, base	2.84	cwt.
Oil, linseed	0.6163	gal.
" coal	0.12	"
Pipe, black 1 in	47.53	M. ft.
" " 2 in	91.84	"
" " 4 in	286.20	"
" " 6 in	516.10	"
" " 8 in	776.87	"
" sewer 6 in	27.94	C. ft.
" " 12 in	52.80	"
" " 24 in	193.00	"
Powder, black	7.1008	cwt.
" "Hercules," 30 per cent. ½" × 8"	11.576 to 12.0754	"
Rails, 60, lb. second-hand	27.50	ton
Rope, Manila, base	10.50	cwt.
Sand and gravel	1.00	ave. cu. yd.
Shafting, 3 7/16 C.R.	3.78	cwt.
Silicate of soda	12.182	bbl.
Steel, structural	3.11	cwt.
Ties, white oak	1.05	each
Tile, 4 × 8 × 12, "El Paso"	58.742	M.
Valves, gate brass "Crane" 1 in	71.96	C.
" " " " 2 in	208.50	"
" " " " 3 in	6.20	each
" " IB " 3 in	4.98	"
Wire, No. 12 weatherproof	23.18	cwt
" No. 4 double braid stranded	56.50	M. ft.
" No. 12 galvanised	6.195	cwt.

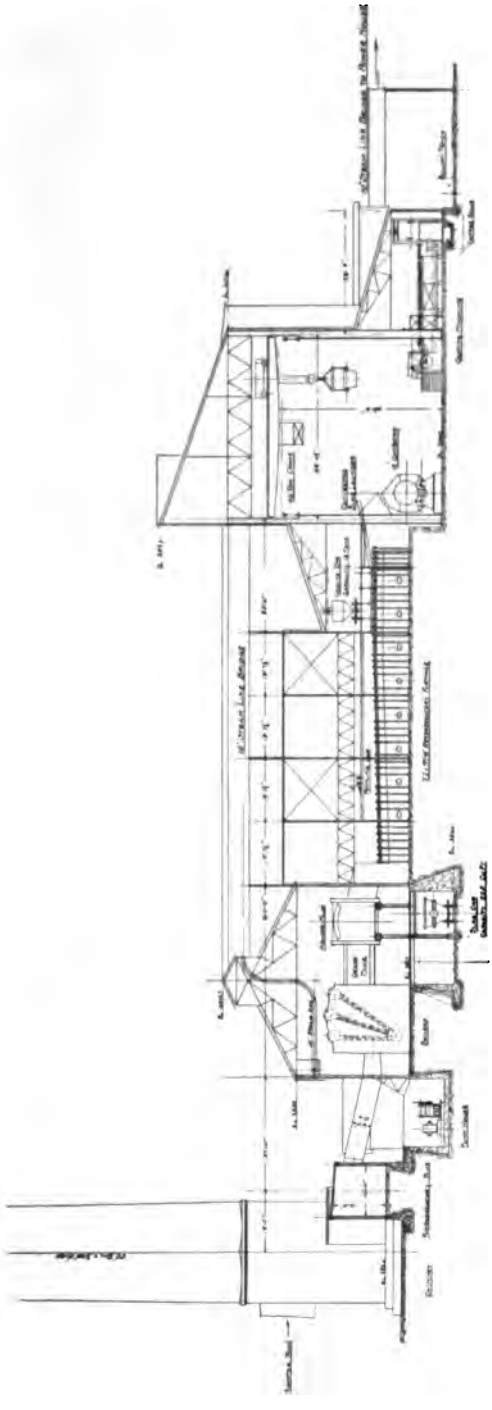


GENERAL PLAN OF SMELTER.
Numbers in circles refer to list on following page.

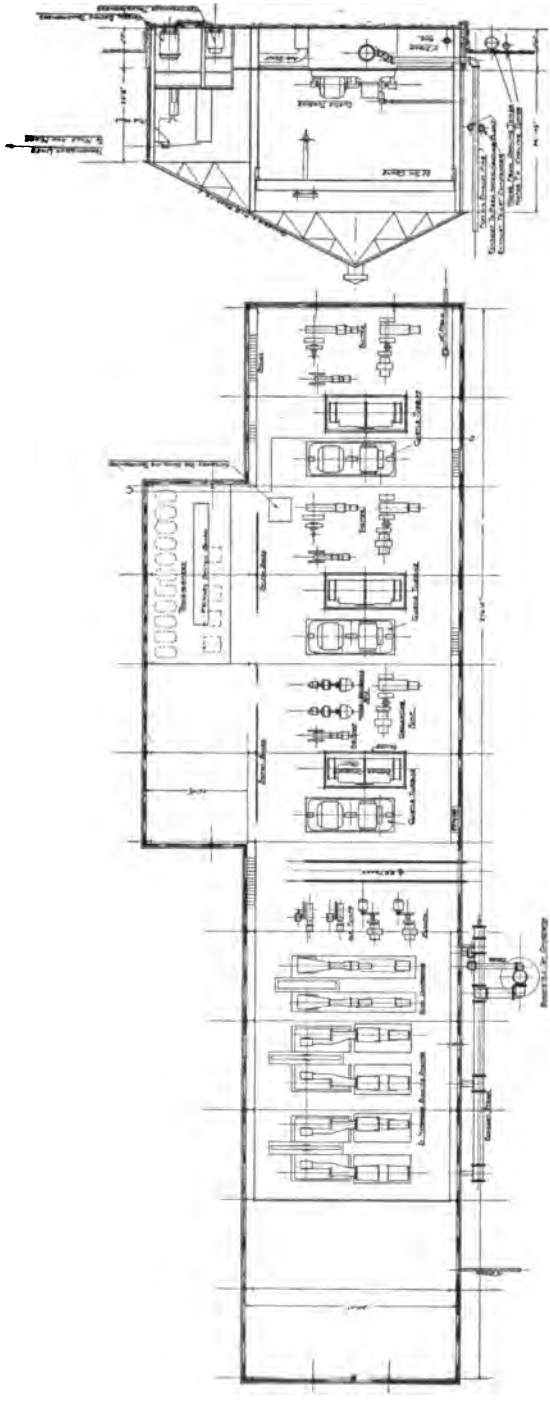
REFERENCE LIST FOR GENERAL PLAN OF SMELTER

(See Preceding Page)

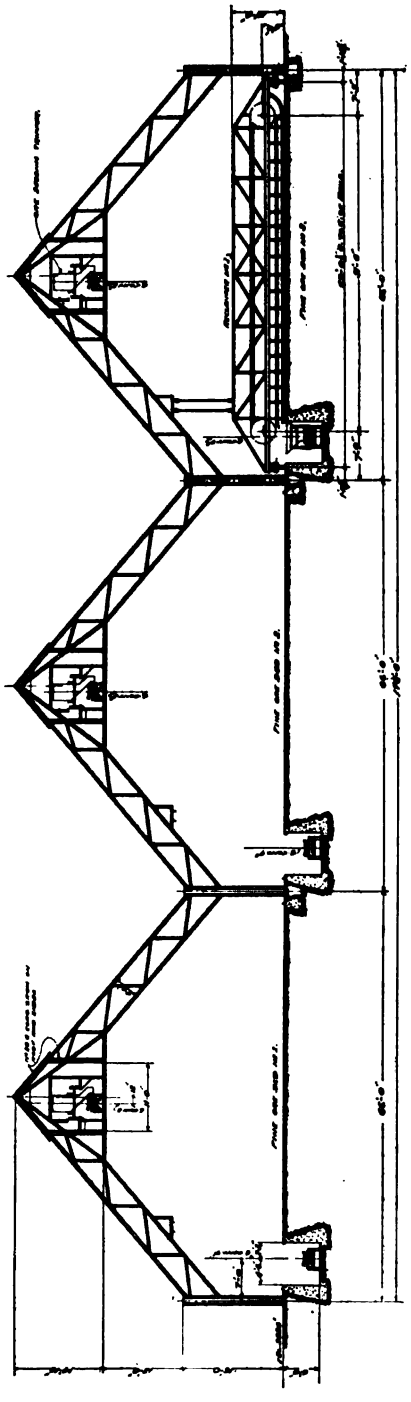
1. Drainage Ditch
2. Depot
3. 6-in. Water Main from Clifton
4. 150-ton Track Scales
5. A. N. M. Main Line
6. 6-in. Water Pipe
7. 250,000-gal. Water Tank
8. Receiving Bins
9. Transfer Pit
10. Crushing Plant
11. Outside Closet
12. Sampling Mill
13. Sample Room
14. Three Spreading Beds
15. Reclaimer House
16. Bunker Bins
17. Roaster Flue
18. Roaster Dust Chamber
19. Roasters
20. 40-ton Track Scale
21. To Slag Dump
22. Calcine Trestle
23. 8-in. Hot-water Line
24. Outside Closet
25. Feed-water Pump House
26. Reverberatory Flue
27. Chimney
28. 163-barrel Oil Tanks
29. Boiler Building
30. Reverberatories
31. Converter Dust Chamber
32. Silica Bins
33. Converter Flue
34. Steam Line
35. Converter Building
36. Converters
37. Casting Shed
38. Bullion Scales
39. Casting Shed
40. 24-in. Blast Pipe
41. Heater House
42. Cooling Tower
43. Machine Shop
44. Boiler Shop
45. Warehouse
46. Culvert
47. Power House
48. Power House Tunnel
49. Condensed Water Pump House
50. Oil Sump
51. 16-in. Oil Pipe
52. Oil Pump House
53. 500,000-gal. Oil Tanks
54. 8-in. Oil Pipe
55. 2 1/2-in. Steam Pipe
56. Assay Office



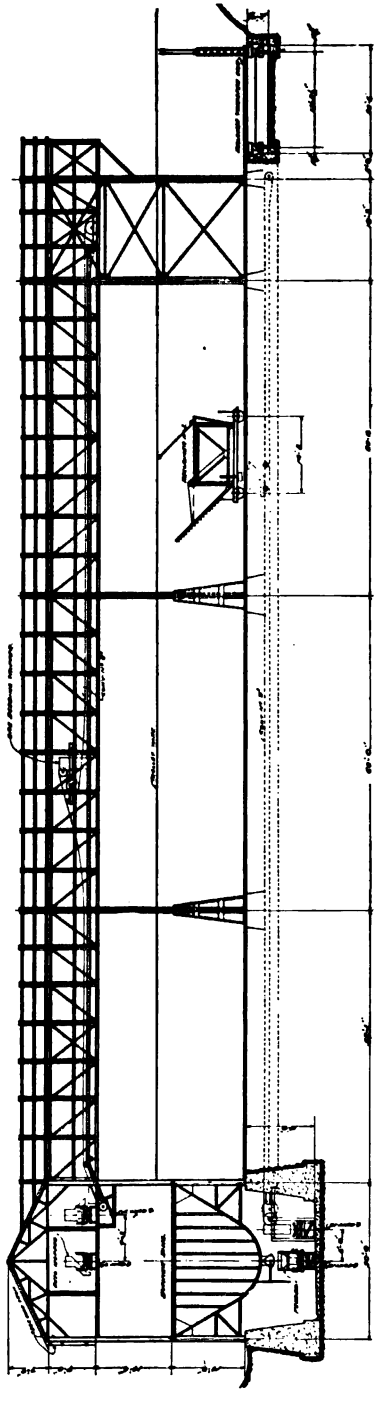
CROSS-SECTION THROUGH REVERBERATORIES AND CONVERTERS.



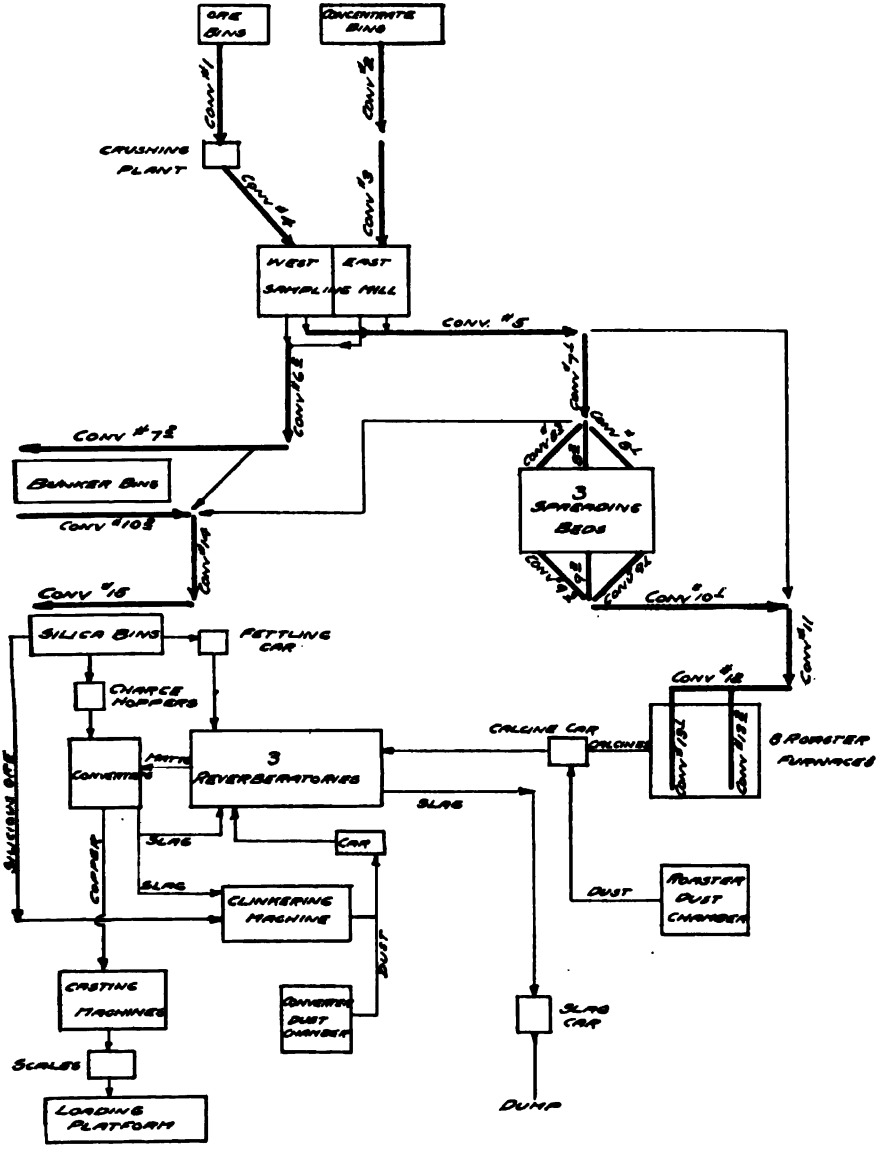
PLAN AND CROSS-SECTION OF POWER HOUSE.



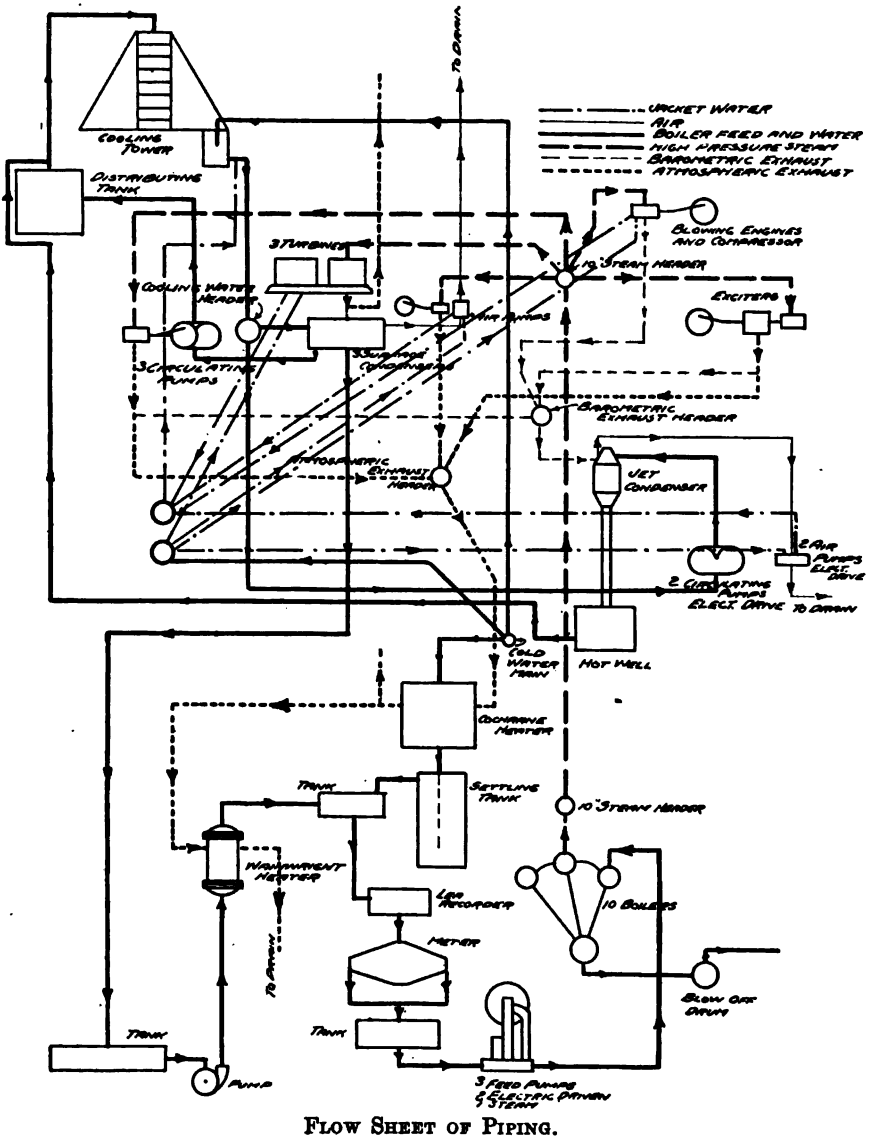
CROSS-SECTION THROUGH BEDDING PLANT, LOOKING EAST.



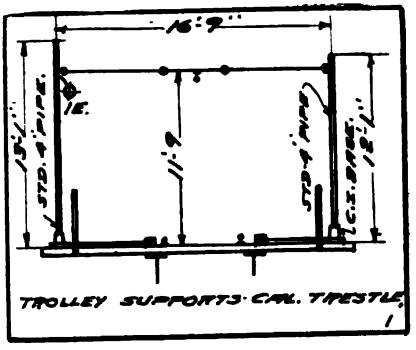
LONGITUDINAL SECTION THROUGH BEDDING PLANT, LOOKING NORTH.



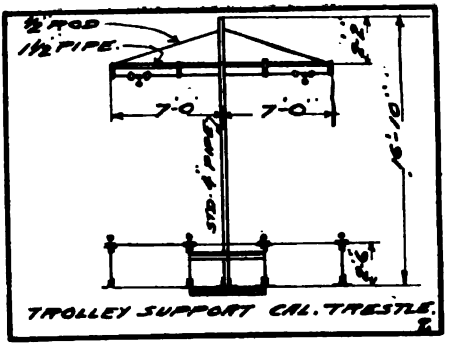
FLOW SHEET OF SMELTER. HEAVY LINES REPRESENT CONVEYORS.



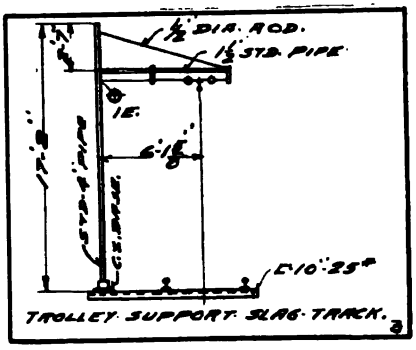
FLOW SHEET OF PIPING.



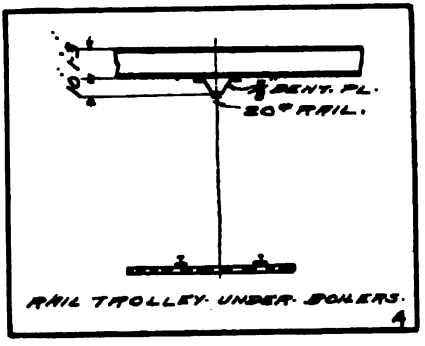
TROLLEY SUPPORTS-CAL. TRESTLE. 1



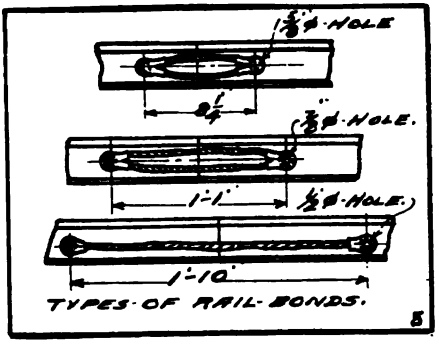
TROLLEY SUPPORT CAL. TRESTLE. 2



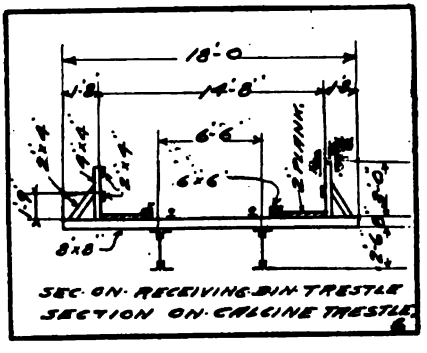
TROLLEY SUPPORT SLAG TRACK. 3



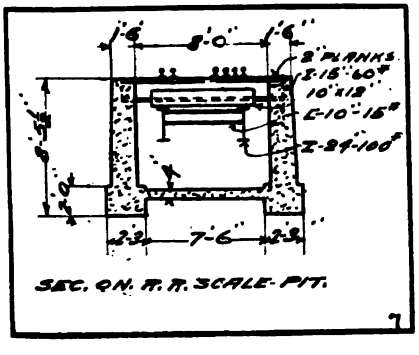
RAIL TROLLEY UNDER BOILERS. 4



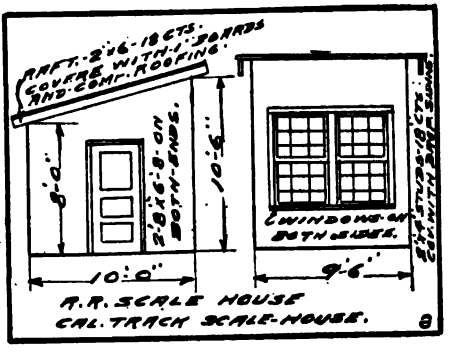
TYPES OF RAIL BONDS. 5



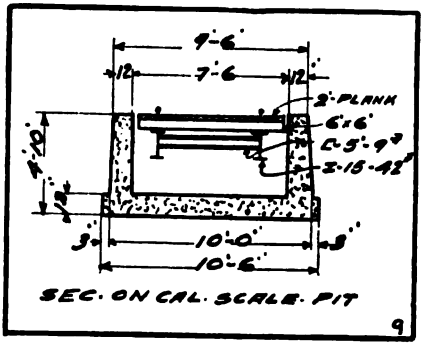
SEC. ON RECEIVING BIN TRESTLE
SECTION ON CALCINE TRESTLE 6



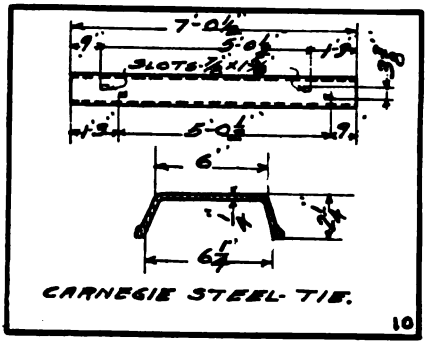
SEC. ON R. R. SCALE PIT. 7



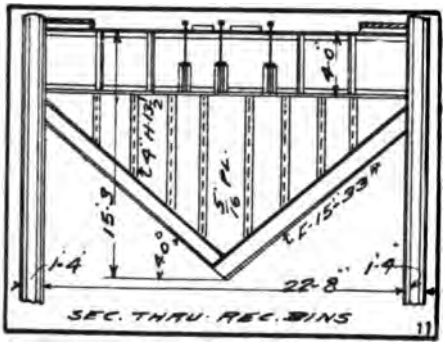
R. R. SCALE HOUSE
CAL. TRACK SCALE-HOUSE. 8



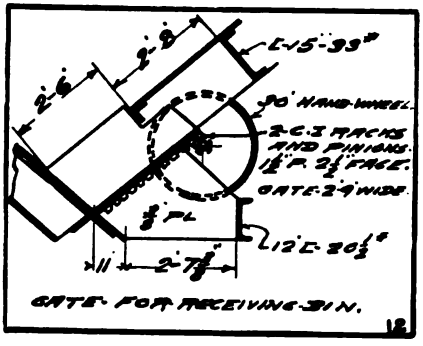
SEC. ON CAL. SCALE PIT 9



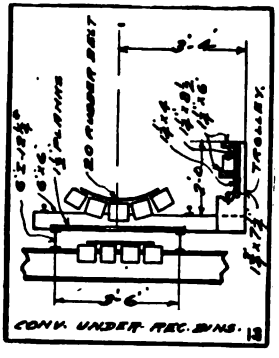
CARNEGIE STEEL TIE. 10



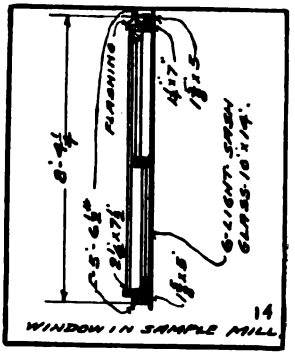
SEC. THRU REC. BINS 11



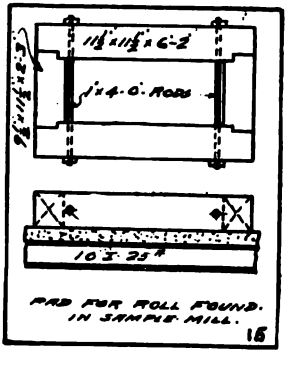
GATE FOR RECEIVING BIN. 12



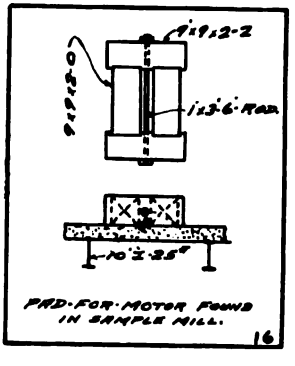
CONV. UNDER REC. BINS. 13



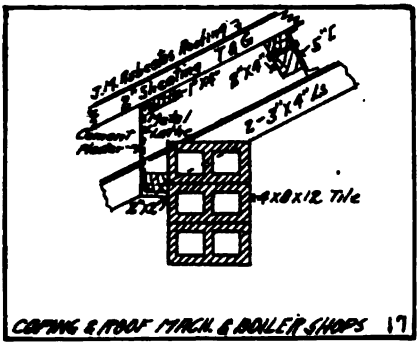
WINDOW IN SAMPLE MILL 14



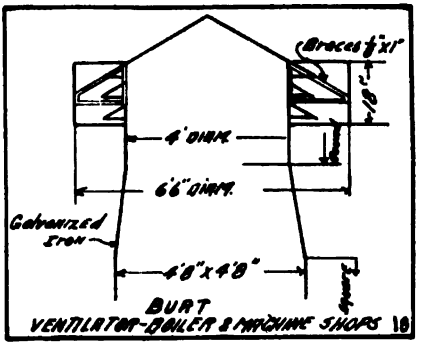
PAD FOR ROLL FOUND. IN SAMPLE MILL. 15



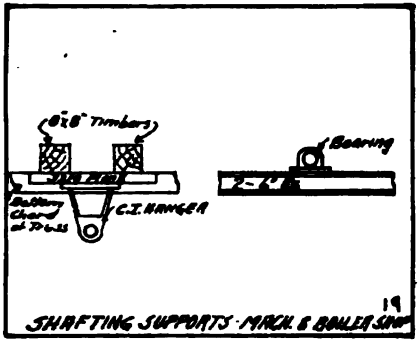
PAD FOR MOTOR FOUND IN SAMPLE MILL. 16



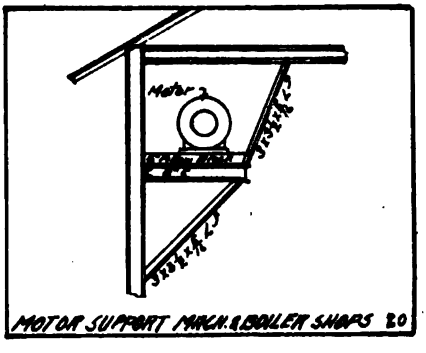
COPING & ROOF MACH. & BOILER SHOPS 17



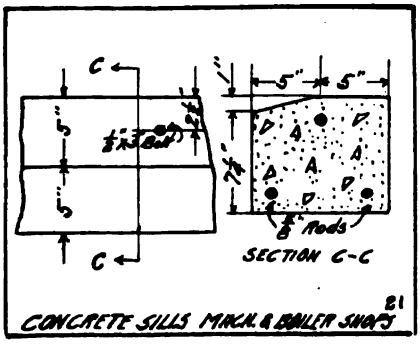
BURT
VENTILATOR-BOILER & MACHINE SHOPS 18



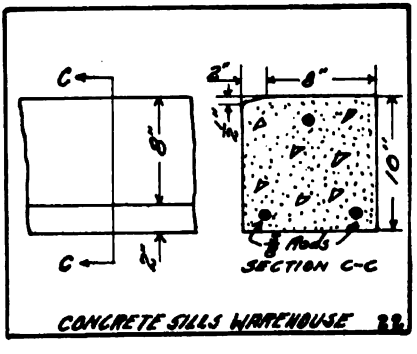
19
SHAFTING SUPPORTS MACH. & BOILER SHOPS



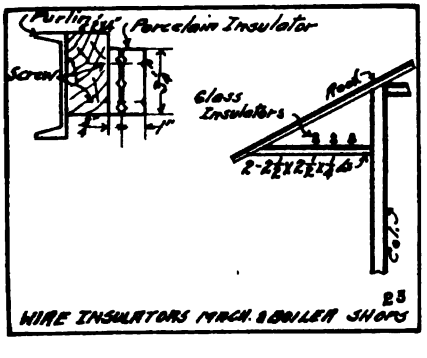
MOTOR SUPPORT MACH. & BOILER SHOPS 20



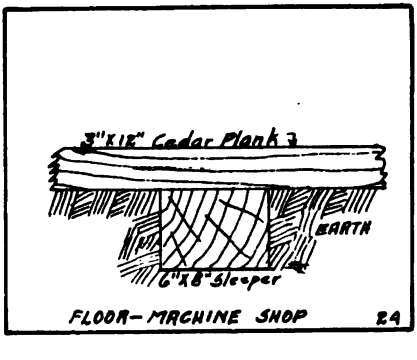
21
CONCRETE SILLS MACH. & BOILER SHOPS



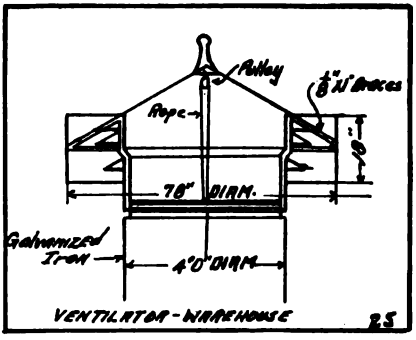
CONCRETE SILLS WAREHOUSE 22



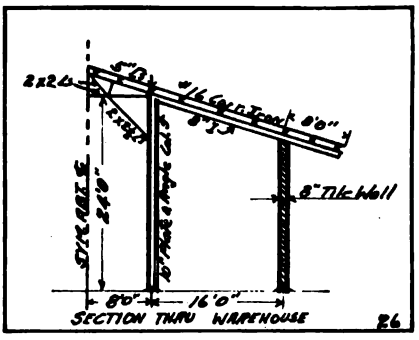
23
WIRE INSULATORS MACH. & BOILER SHOPS



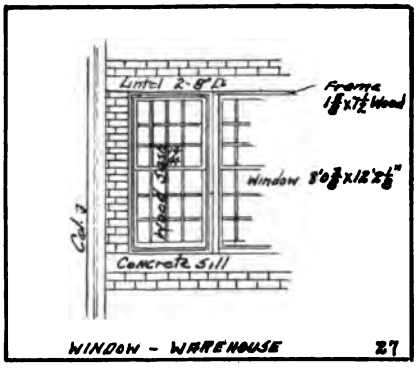
FLOOR—MACHINE SHOP 24



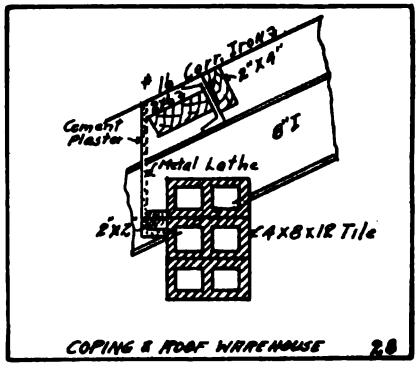
VENTILATOR - WAREHOUSE 25



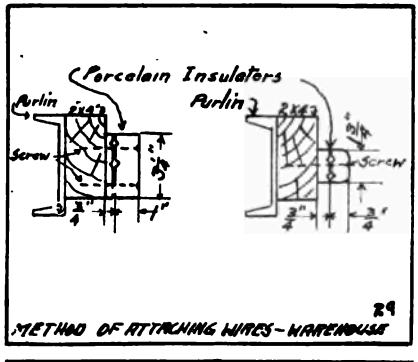
SECTION THRU WAREHOUSE 26



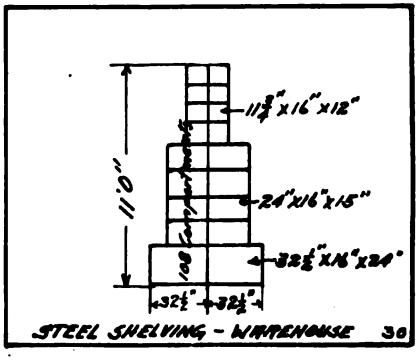
WINDOW - WAREHOUSE 27



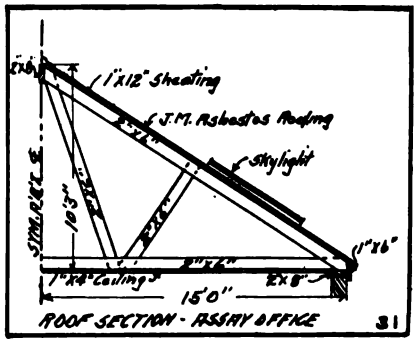
COPING & ROOF WAREHOUSE 28



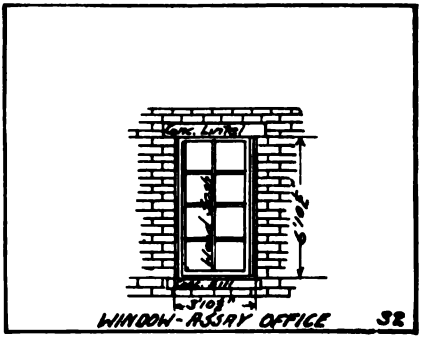
29
METHOD OF ATTACHING WIRES - WAREHOUSE



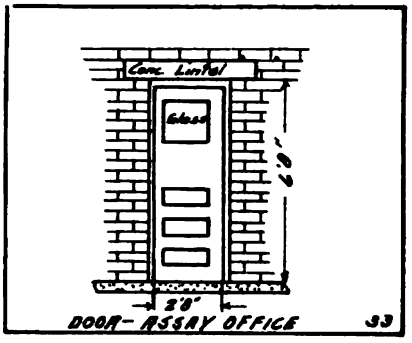
STEEL SHELVING - WAREHOUSE 30

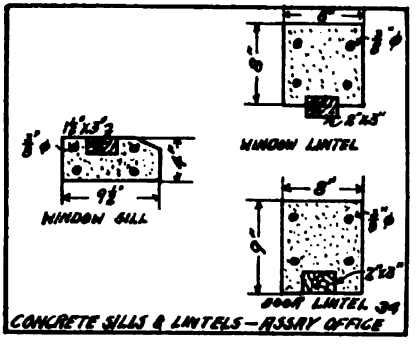


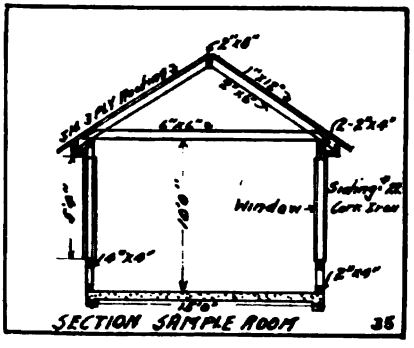
ROOF SECTION - ASSAY OFFICE 31

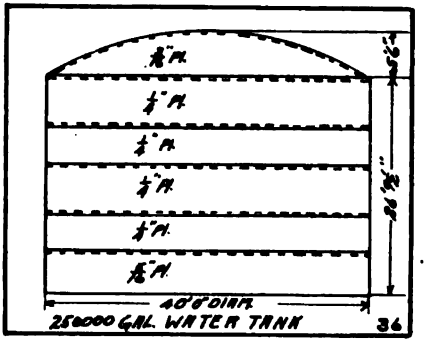


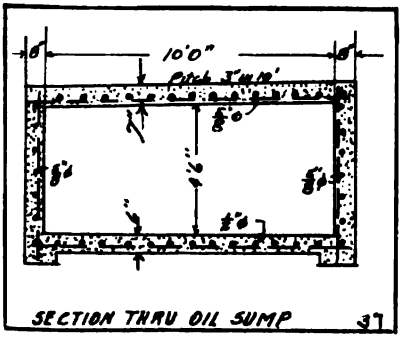
WINDOW - ASSAY OFFICE 32

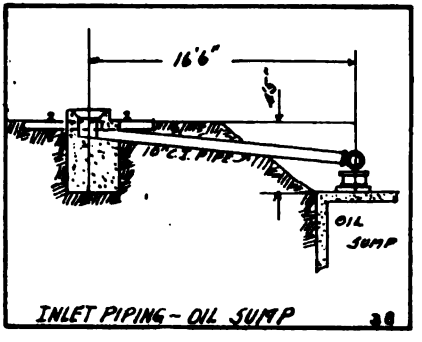

Conc. Lintel
Glass
2'8"
DOOR - ASSAY OFFICE 33

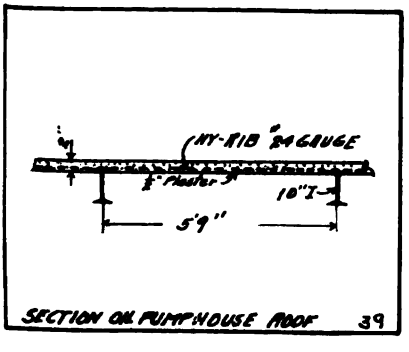

WINDOW LINTEL
WINDOW SILL
DOOR LINTEL 34
CONCRETE SILLS & LINTELS - ASSAY OFFICE

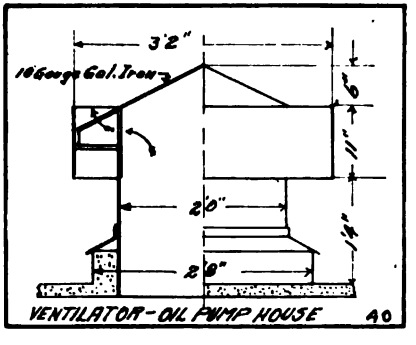

Window
SECTION SAMPLE ROOM 35

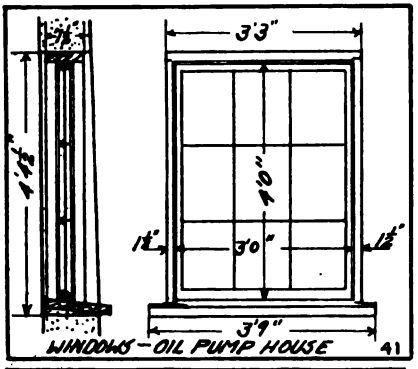

40'8" DIAM.
25000 GAL. WATER TANK 36


10'0"
SECTION THRU OIL SUMP 37

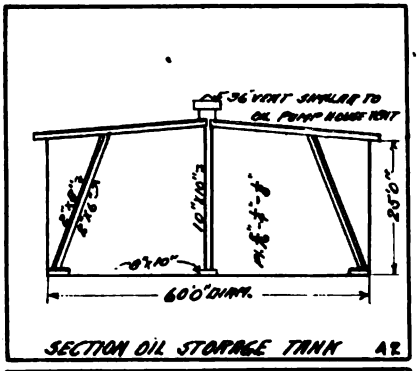

16'6"
OIL SUMP
INLET PIPING - OIL SUMP 38


5'9"
SECTION OIL PUMPHOUSE ROOF 39

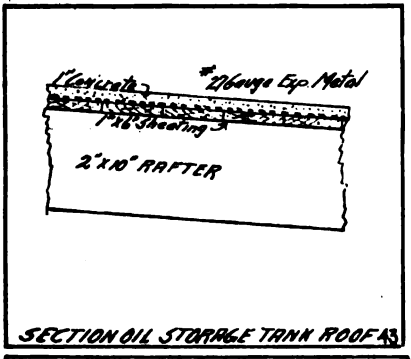

3'2"
2'0"
2'8"
VENTILATOR - OIL PUMP HOUSE 40



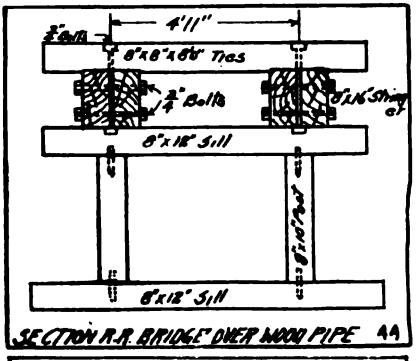
WINDOWS – OIL PUMP HOUSE 41



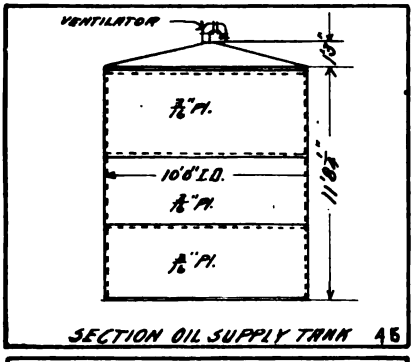
SECTION OIL STORAGE TANK 42



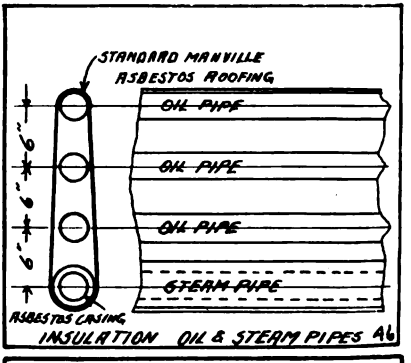
SECTION OIL STORAGE TANK ROOF 43



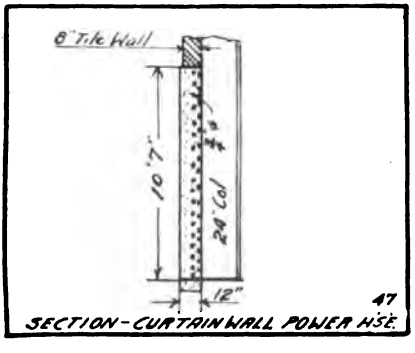
SECTION R.R. BRIDGE OVER WOOD PIPE 44



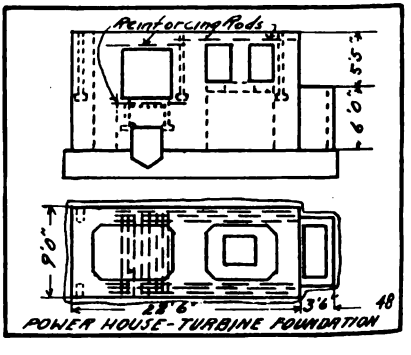
SECTION OIL SUPPLY TANK 45



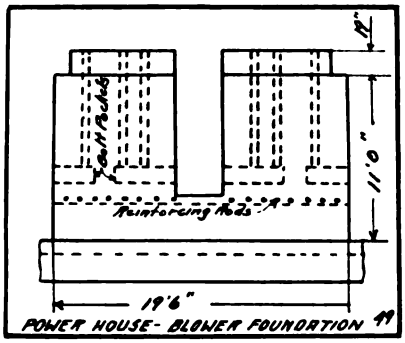
INSULATION OIL & STEAM PIPES 46



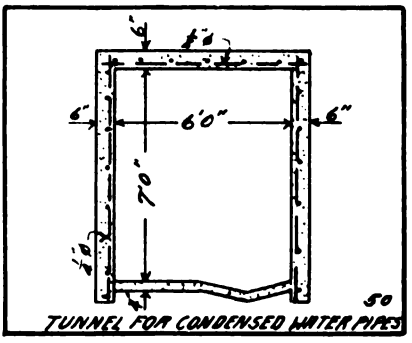
47
SECTION – CURTAIN WALL POWER HSE.



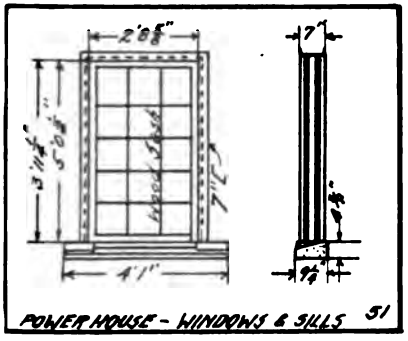
48
POWER HOUSE – TURBINE FOUNDATION



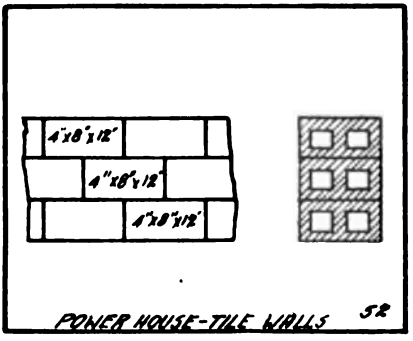
POWER HOUSE - BLOWER FOUNDATION 49



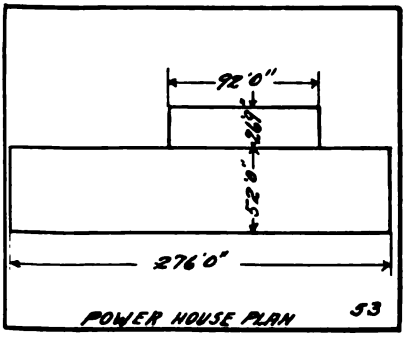
50 TUNNEL FOR CONDENSED WATER PIPES



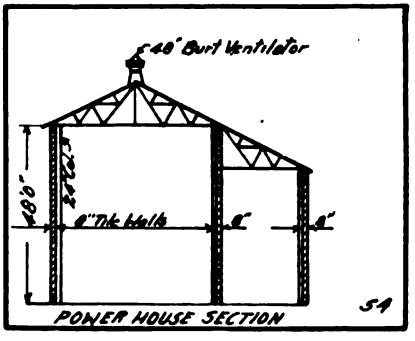
POWER HOUSE - WINDOWS & SILLS 51



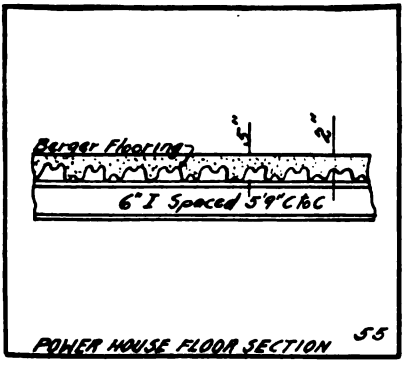
POWER HOUSE - TILE WALLS 52



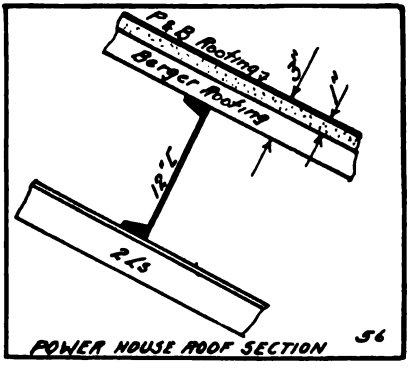
POWER HOUSE PLAN 53



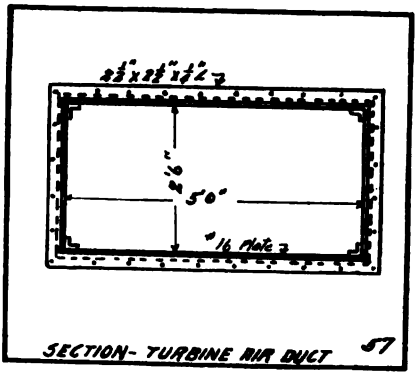
POWER HOUSE SECTION 54



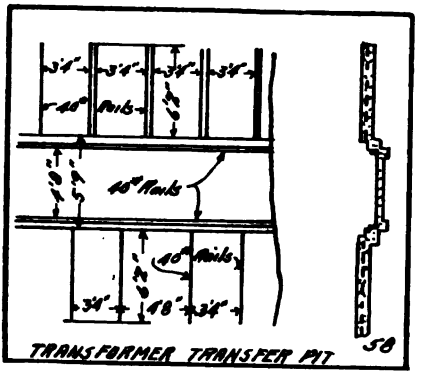
POWER HOUSE FLOOR SECTION 55



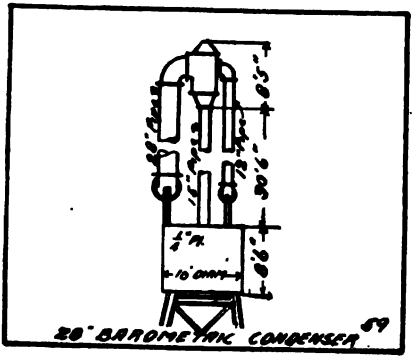
POWER HOUSE ROOF SECTION 56



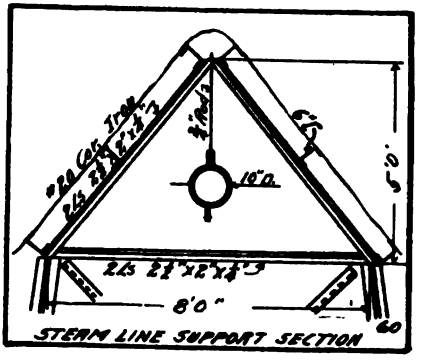
SECTION- TURBINE AIR DUCT 57



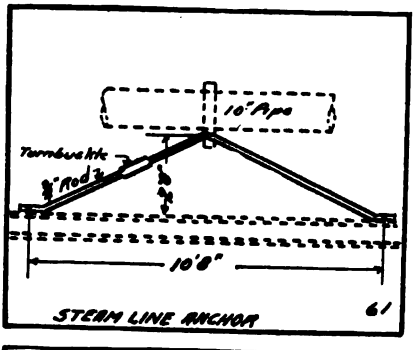
TRANSFORMER TRANSFER PIT 58



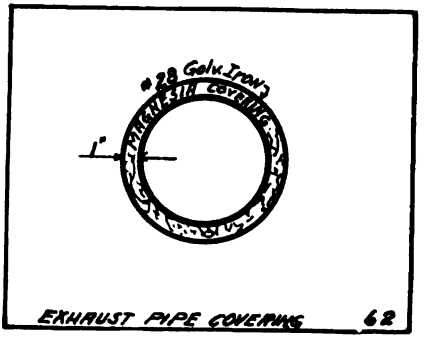
28" BAROMETRIC CONDENSER 59



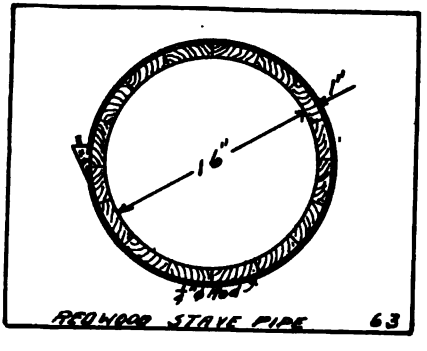
STEAM LINE SUPPORT SECTION 60



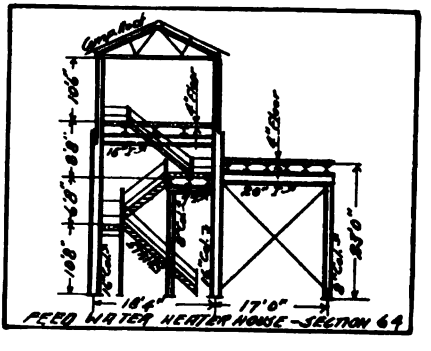
STEAM LINE ANCHOR 61



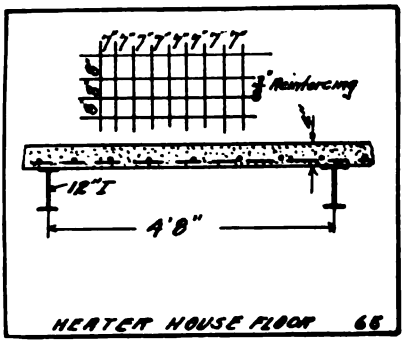
EXHAUST PIPE COVERING 62



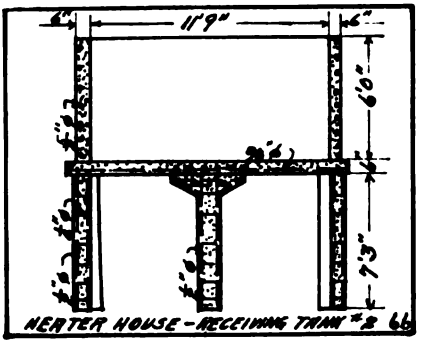
REDWOOD STAVE PIPE 63



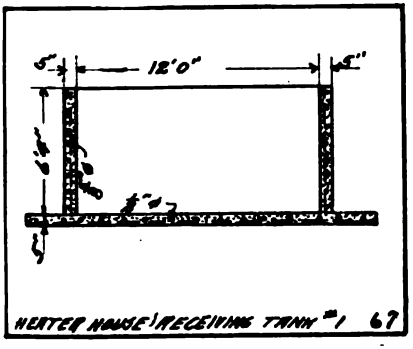
FEED WATER HEATER HOUSE - SECTION 64



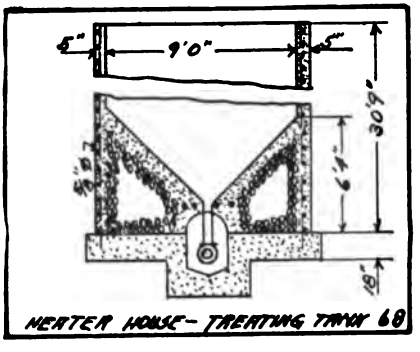
HEATER HOUSE FLOOR 66



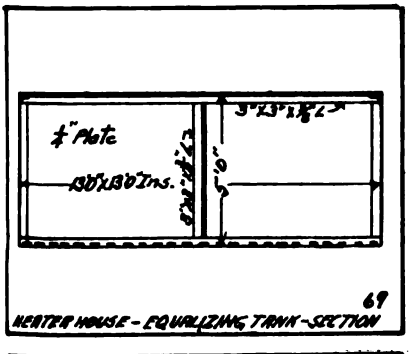
HEATER HOUSE - RECEIVING TANK #2 65



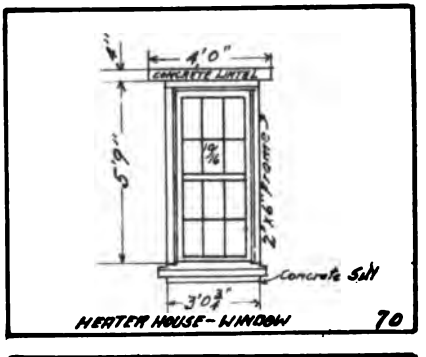
HEATER HOUSE RECEIVING TANK #1 67



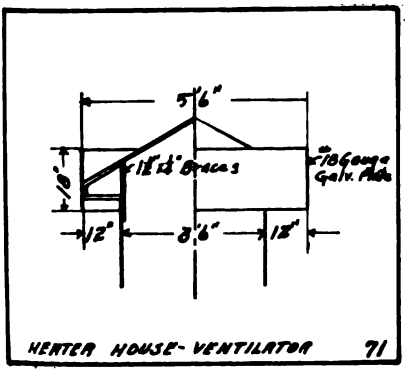
HEATER HOUSE - TREATING TANK 68



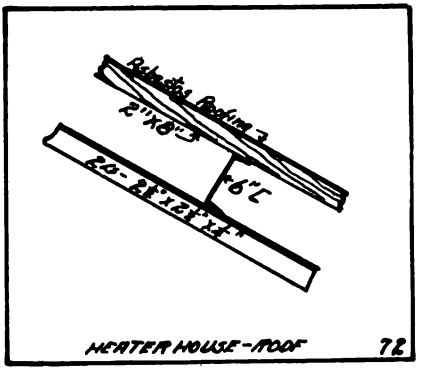
69
HEATER HOUSE - EQUALIZING TANK - SECTION



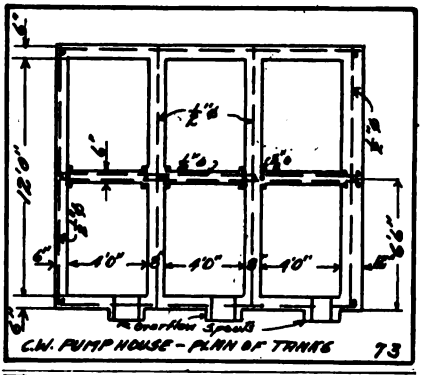
HEATER HOUSE - WINDOW 70



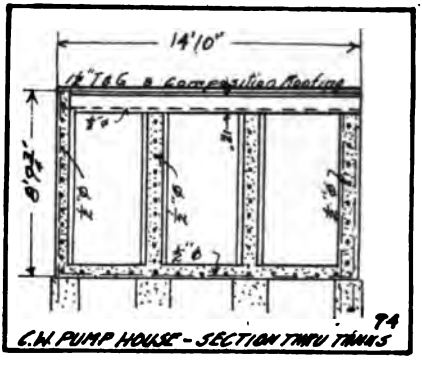
HEATER HOUSE - VENTILATOR 71



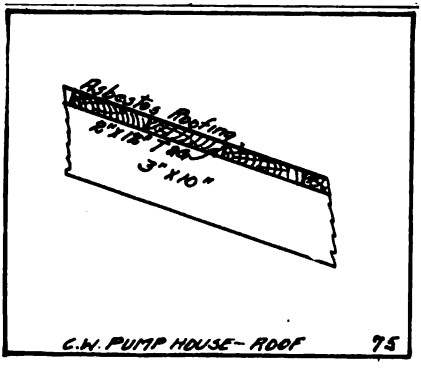
HEATER HOUSE - ROOF 72



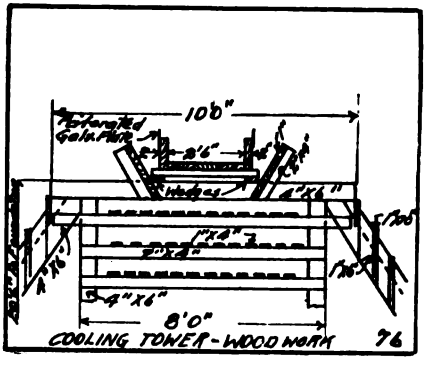
C.W. PUMP HOUSE - PLAN OF TANKS 73



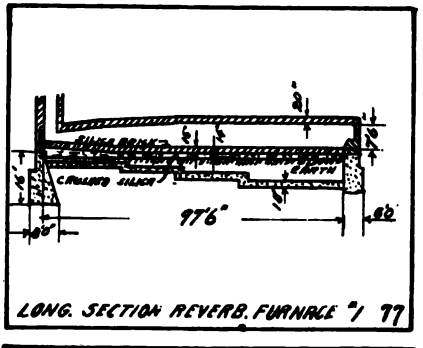
C.W. PUMP HOUSE - SECTION THRU TANKS 74



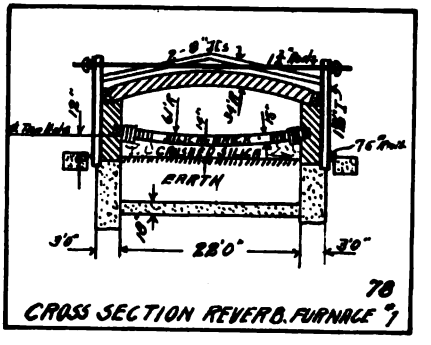
C.W. PUMP HOUSE - ROOF 75



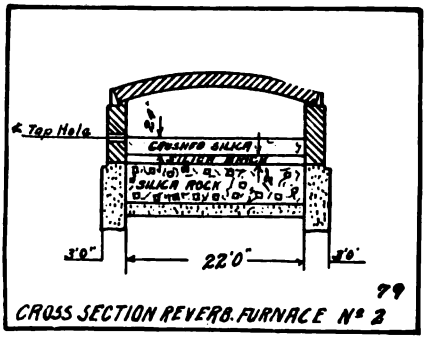
COOLING TOWER - WOOD WORK 76



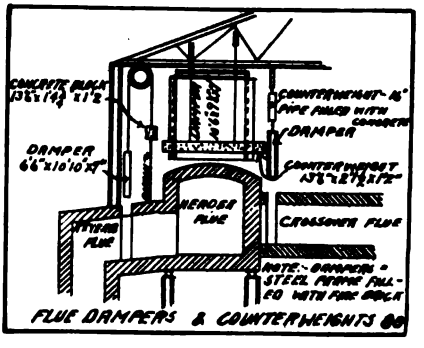
LONG. SECTION REVERB. FURNACE "1 77



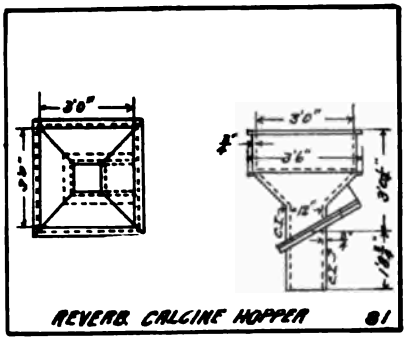
78
CROSS SECTION REVERB. FURNACE "1

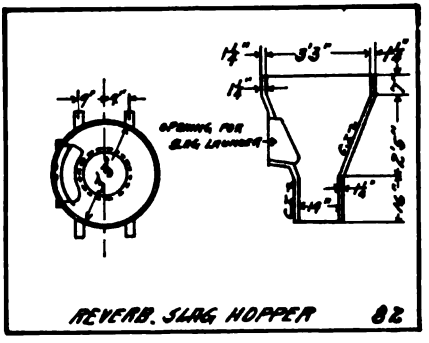


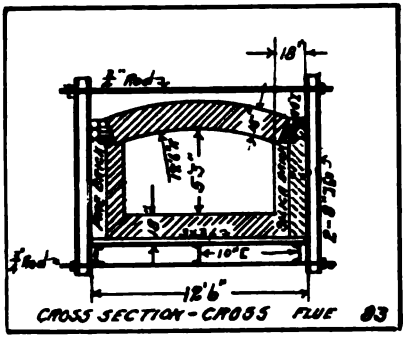
79
CROSS SECTION REVERB. FURNACE Nº 2

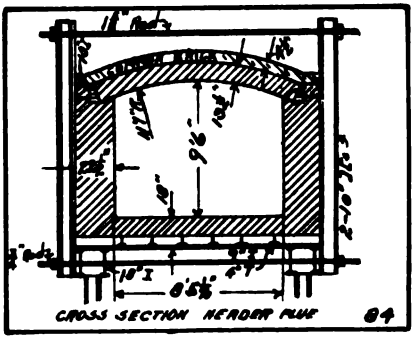


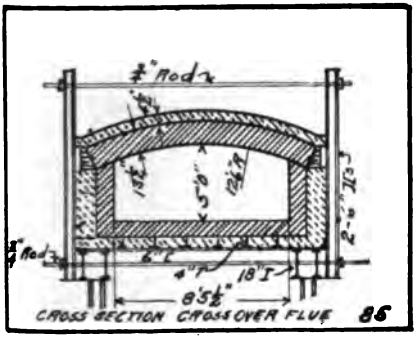
FLUE DAMPERS & COUNTERWEIGHTS 80

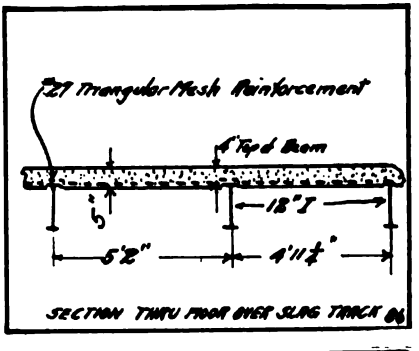

3'0"
3'6"
REVERB. CALCINE HOPPER 81

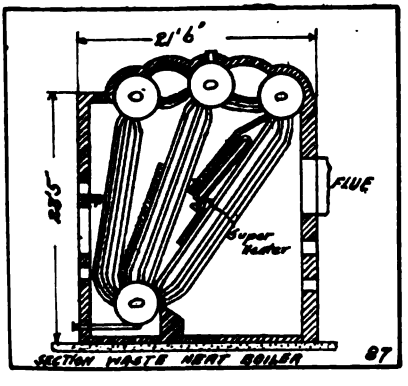

3'3"
OPENING FOR SLAG LADDER
REVERB. SLAG HOPPER 82

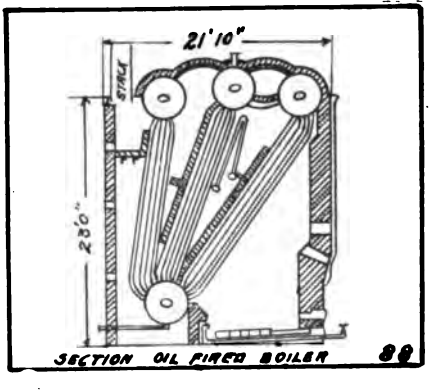

18"
12'6"
CROSS SECTION - CROSS FLUE 83

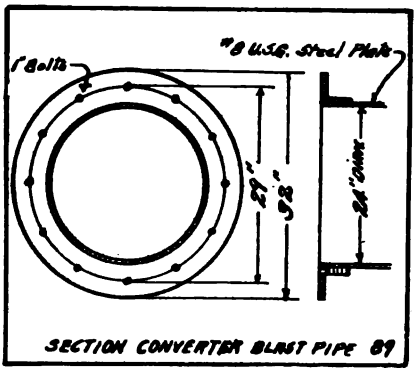

9'6"
8'5½"
CROSS SECTION HEADER FLUE 84


5'0"
8'5½"
CROSS SECTION CROSSOVER FLUE 85

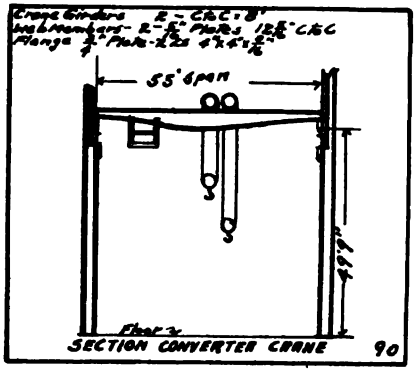

27 Triangular Mesh Reinforcement
5"
5'2"
4'11½"
SECTION THRU FLOOR OVER SLAG TANK 86


21'6"
FLUE
Super Heater
SECTION WASTE HEAT BOILER 87

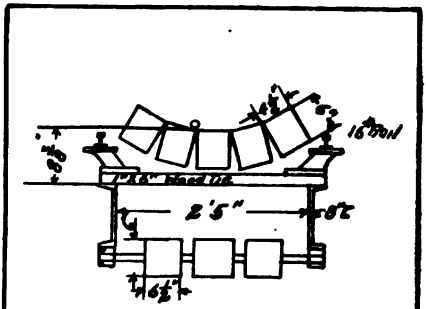

21'10"
STACK
230"
SECTION OIL FIRED BOILER 88



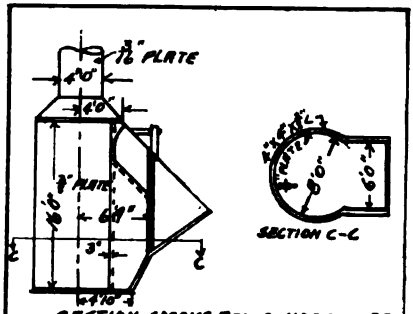
SECTION CONVERTER BLAST PIPE 89



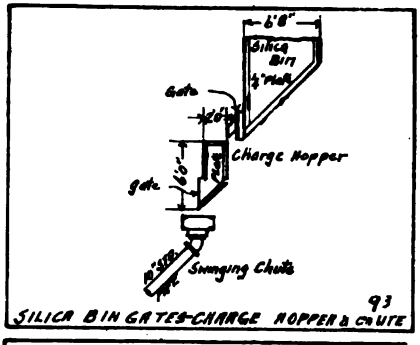
SECTION CONVERTER CRANE 90



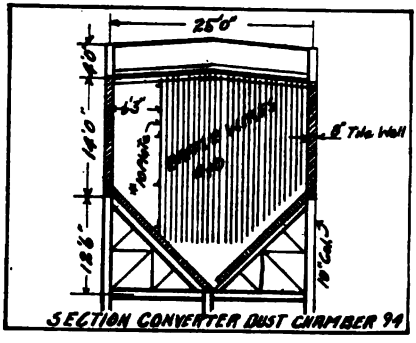
SECTION CONVEYORS 15, 13-1, 13-2, 7-1, 7-2 91



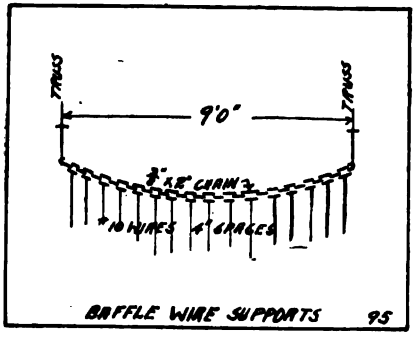
SECTION SMOKE BOX & HOOD 92



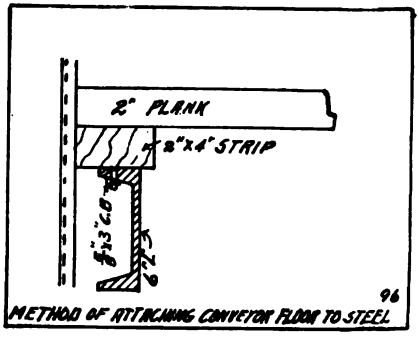
SILICA BIN GATES-CHARGE HOPPER & CHUTE 93



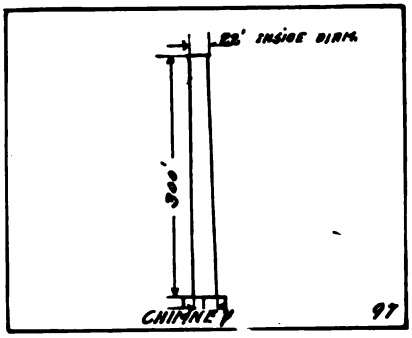
SECTION CONVERTER DUST CHAMBER 94



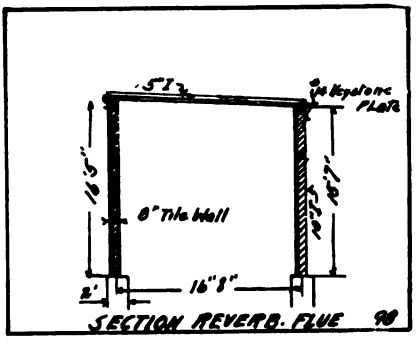
BAFFLE WIRE SUPPORTS 95



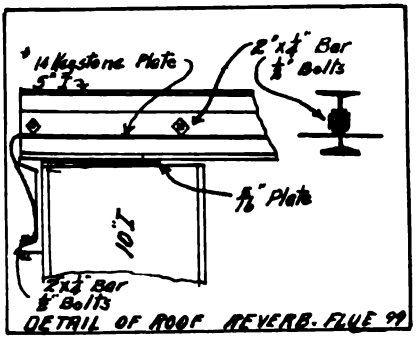
METHOD OF ATTACHING CONVEYOR FLOOR TO STEEL 96



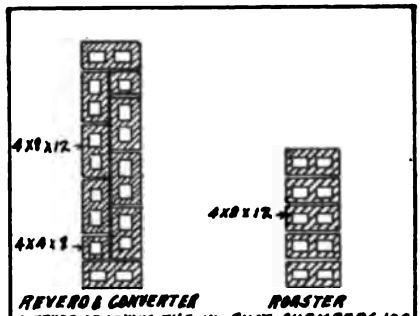
CHIMNEY 97



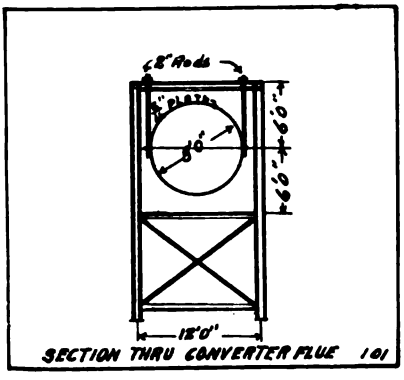
SECTION REVERB. FLUE 98



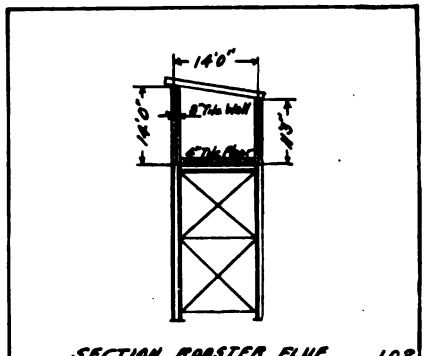
DETAIL OF ROOF REVERB. FLUE 99



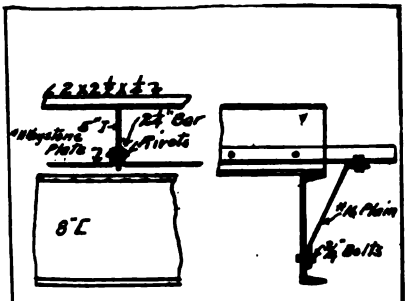
REVERB & CONVERTER ROASTER
METHOD OF LAYING TILE IN DUST CHAMBERS 100



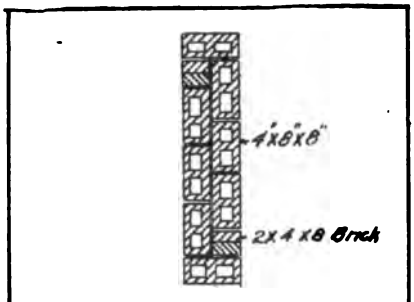
SECTION THRU CONVERTER FLUE 101



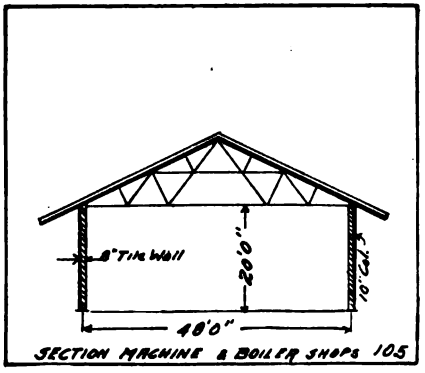
SECTION ROASTER FLUE 102



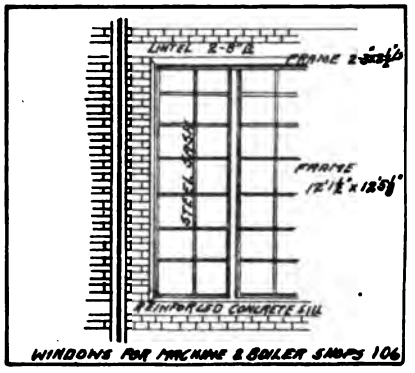
DETAIL OF ROOF ROASTER FLUE 103



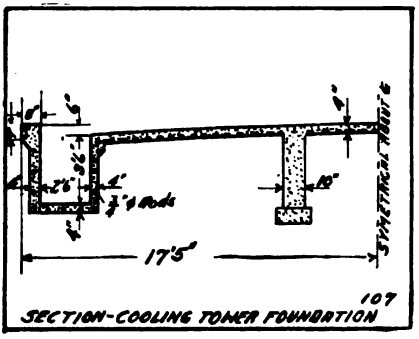
104
METHOD OF LAYING TILE IN ROASTER FLUE



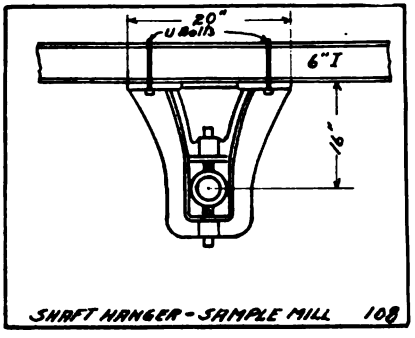
SECTION MACHINE & BOILER SHOPS 105



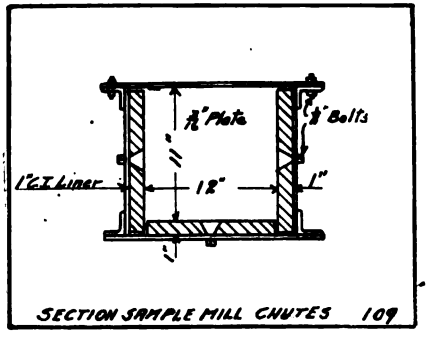
WINDOWS FOR MACHINE & BOILER SHOPS 106



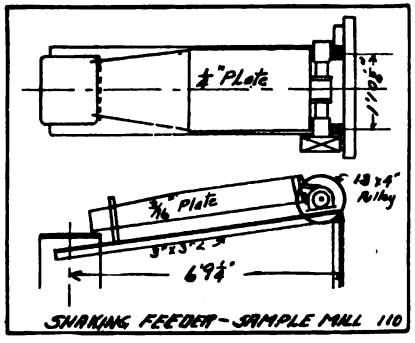
SECTION-COOLING TOWER FOUNDATION 107



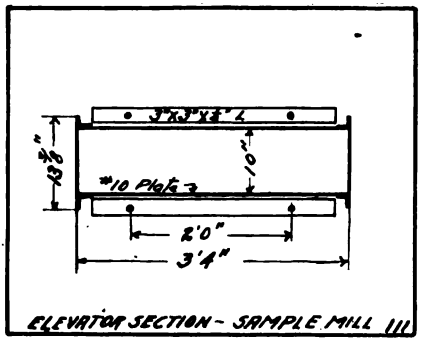
SHAFT HANGER - SAMPLE MILL 108



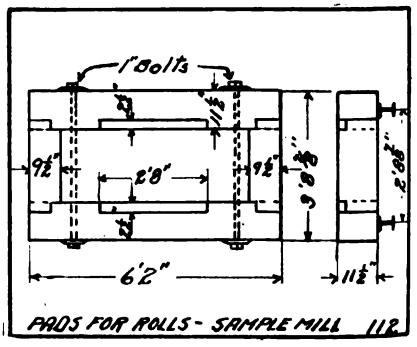
SECTION SAMPLE MILL CHUTES 109



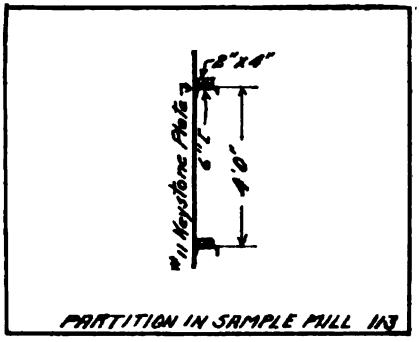
SHAKING FEEDER - SAMPLE MILL 110



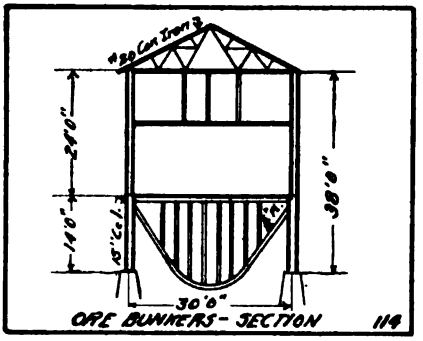
ELEVATOR SECTION - SAMPLE MILL 111



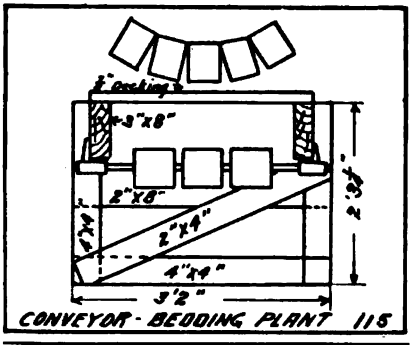
PADS FOR ROLLS - SAMPLE MILL 112



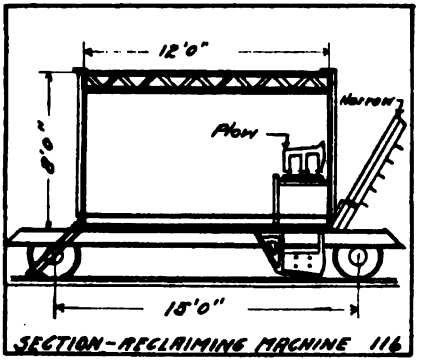
PARTITION IN SAMPLE MILL 113



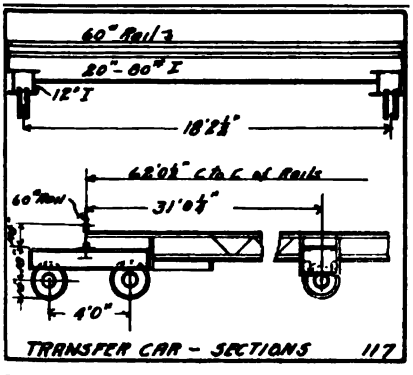
ORE BUNKERS - SECTION 114



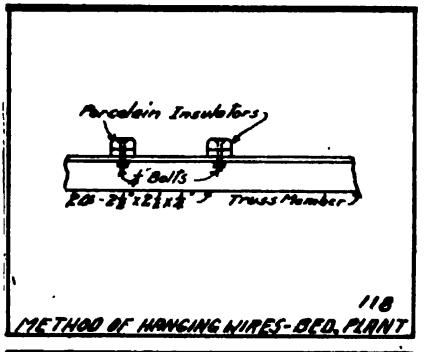
CONVEYOR - BEDDING PLANT 115



SECTION - RECLAIMING MACHINE 116

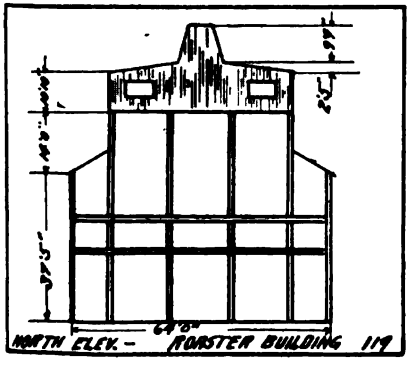


TRANSFER CAR - SECTIONS 117

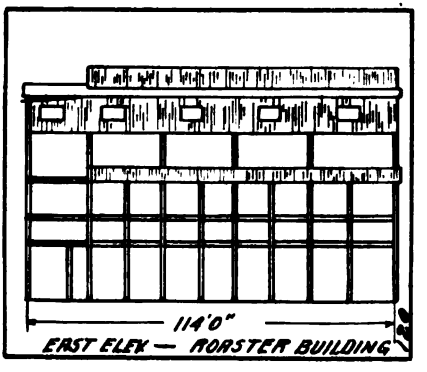


118

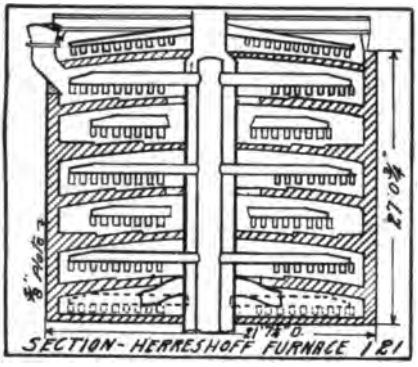
METHOD OF HANGING WIRES - BED. PLANT

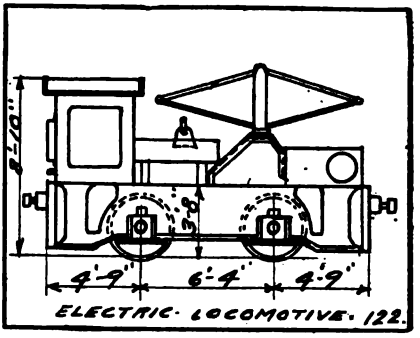


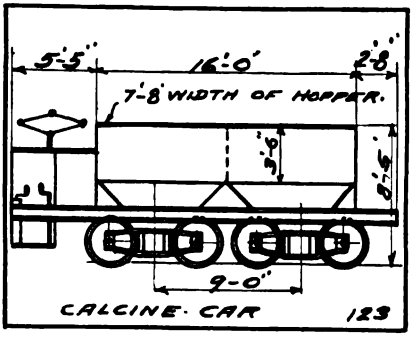
NORTH ELEV. - ROASTER BUILDING 119

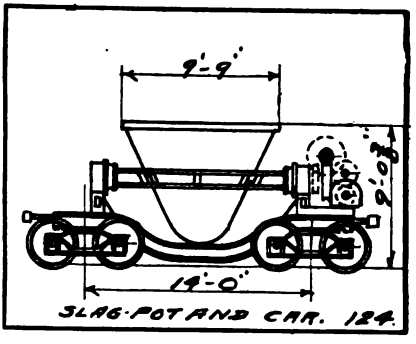


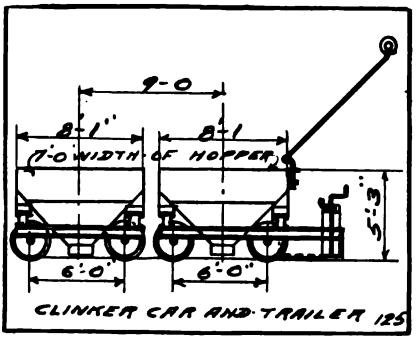
EAST ELEV — ROASTER BUILDING 120

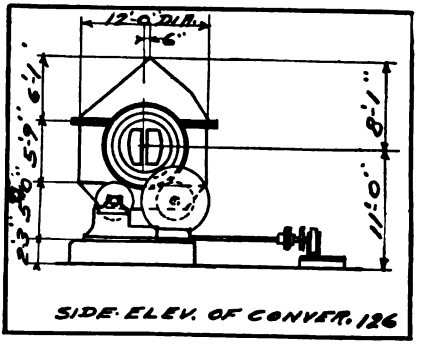

27'0"
21'4"
SECTION-HERRESHOFF FURNACE 121

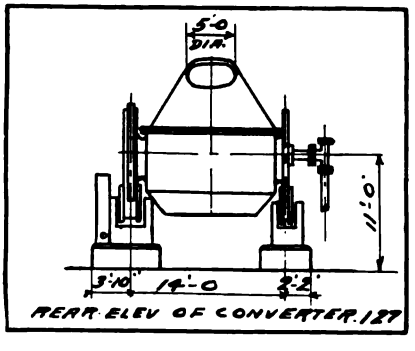

8'10"
3'8"
4'9"
6'4"
4'9"
ELECTRIC LOCOMOTIVE. 122.

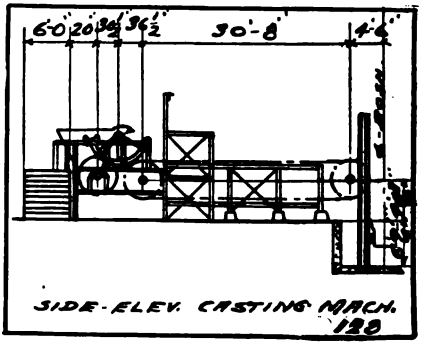

5'5"
16'0"
2'8"
7'8" WIDTH OF HOPPER.
3'6"
8'6"
9'0"
CALCINE CAR 123

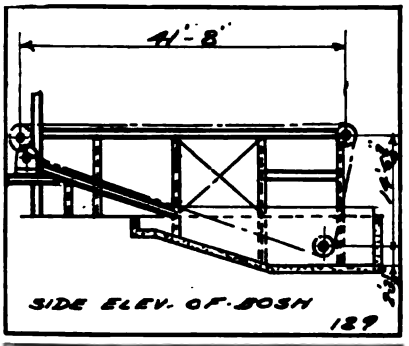

9'9"
9'0"
14'0"
SLAG POT AND CAR. 124.

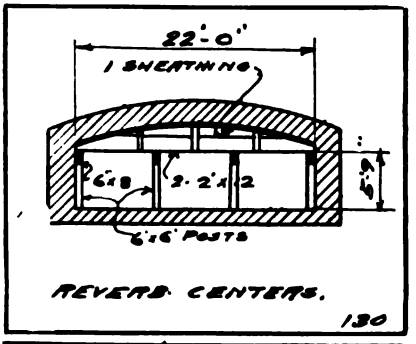

9'0"
8'1"
8'1"
7'0" WIDTH OF HOPPER
5'3"
6'0"
6'0"
CLINKER CAR AND TRAILER 125

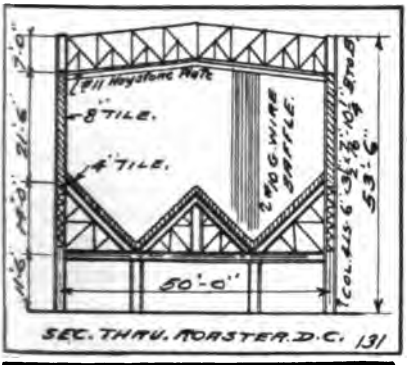

12'0" DIA.
6"
8'1"
11'0"
SIDE ELEV. OF CONVER. 126

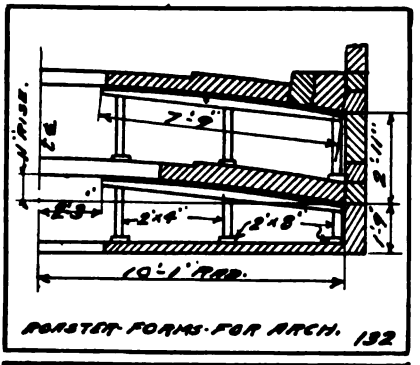

5'0"
DIA.
11'0"
3'10"
14'0"
REAR ELEV OF CONVERTER 127

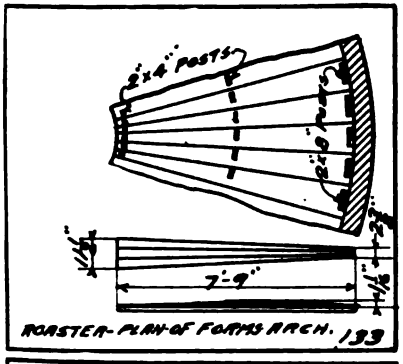

6'0"
30'8"
4'6"
SIDE ELEV. CASTING MACH. 128

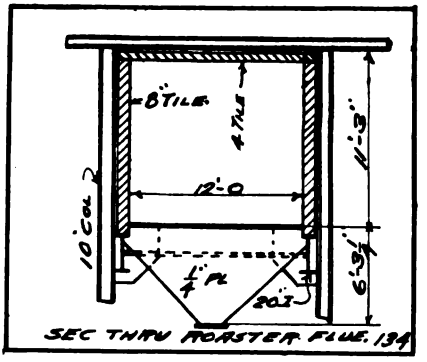

4'-8"
SIDE ELEV. OF BOSH
129

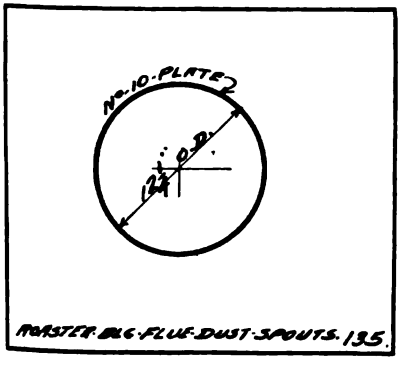

22'-0"
1 SHEATHING
6"x8"
2-2"x12"
6"x6" POSTS
REVERB. CENTERS.
130

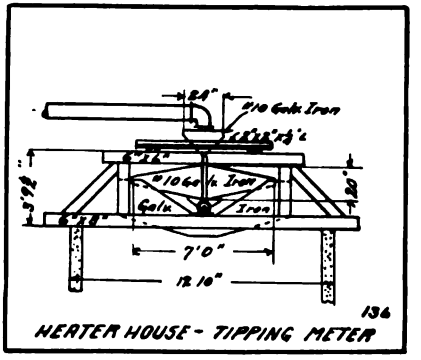

8" TILE.
4" TILE.
50'-0"
SEC. THRU. ROASTER D.C.
131


7'-9"
10'-1" REF.
ROASTER FORMS FOR ARCH.
132


2"x4" POSTS
7'-9"
ROASTER PLAN OF FORMS ARCH.
133


8" TILE.
12'-0"
11'-3"
6'-3"
SEC THRU ROASTER FLUE.
134


No. 10 PLATE
ROASTER BIG FLUE DUST SPOUTS.
135


7'0"
12'10"
136
HEATER HOUSE - TIPPING METER

CHAPTER VI

DESCRIPTION OF COSTS

ENGINEERING

Account 7100—Total Engineering Expense.

This account is a summation of accounts 7,101 to 7,206 inclusive, also of 9,000.1, power plant engineering expense. These accounts cover the engineering expenditures as their respective headings would indicate. As the total engineering expense they represent a percentage of the total cost of the smelter, less the engineering and indirect expenses, and have been so reported. In the making of any total estimate based on the unit costs derived from this sheet, it is assumed that of the total estimate so arrived at, 5.40 per cent. will be taken and added thereto to ascertain the item of engineering.

Account 7001—See Account 8999.
Account 7004—See Account 8999.
Account 7101—See Account 7100.
Account 7103—See Account 7100.
Account 7104—See Account 7100.
Account 7105—See Account 7100.
Account 7201—See Account 7100.
Account 7202—See Account 7100.
Account 7203—See Account 7100.
Account 7204—See Account 7100.
Account 7205—See Account 7100.
Account 7206—See Account 7100.

YARD TRACKS AND INDUSTRIAL SYSTEM

Tracks

Account 7301—Excavation.

This cost covers all of the excavating, barrow and grading incident to bringing the road beds of the New Smelter tracks to sub-grade. The materials worked varied from rock (Gila conglomerate) through hard clayey soil filled with one-man stones, to light loam. The means employed to excavate covered the use of powder, plows, picks and shovels, slips and fresnos. The work covered by this cost was not carried on continuously, but as conditions about the plant permitted. The unit cost represents fairly the average cost of shallow excavating in large amounts about the smelter site.

Account 7302—Ties.

This account represents 700 steel ties, 7,656 white-oak ties laid in track, 524 white-oak ties in stock, 300 Oregon pine switch ties laid in track, and 1,082 white-oak ties used in temporary tracks and in handling machinery. There is also included here the labor of unloading, stocking, handling to points about the plant and placing upon the various grades. The steel ties are 7 ft. long, furnished with four No. 23 clips for 60-lb. A. S. C. E. rail, laid on 4 ft. 8½ in. gauge. They

were furnished by the Carnegie Steel Co. and cost $1.66 each. (See Fig. 10.) The white-oak ties are rough hewn, 6 in. by 8 in. by 8 ft., and cost $1.05 apiece.

The switch ties were of Oregon pine, standard size and various lengths, according to their position in the track and the size of frog. They cost at the rate of $30 a thousand, board measure. The account thus stands as follows:

Labor	$425.13	
Wood ties	9,614.86	
Steel ties	1,162.93	
		$11,202.92

The average cost of each wood tie in the track represents $1.31. The ties were laid about 58 to the 100 ft. (See Fig. 10.)

Account 7303—**Rails and Rail Fastenings.**

The rails covered by this account were second hand, purchased from the Arizona and New Mexico Railway; 60 lb. A. S. C. E. standard. The track laid totaled 14,813 ft. long and was divided as follows:

On wood ties in dirt		On trestle	
Track No. 6	2,660 ft.	Track No. 7	294 ft.
7	1,023 ft.	15	403 ft.
8	1,940 ft.		
9	340 ft.	On steel ties	
10	645 ft.	Track No. 13	665 ft.
12	1,053 ft.	14	283 ft.
13	1,396 ft.		
14	1,060 ft.		
15	1,685 ft.		
17	212 ft.		
18	279 ft.		
19	290 ft.		
24	585 ft.		

The account is divided up as follows:

29,626 ft. 60 lb. rail @ $27.50 a ton	$8,147.15	
986 pr. angle bars @ $0.50	493.00	
4,000 bolts @ $3.50 a hundred	140.00	
4,000 nut locks @ $1.07 a hundred	42.80	
95 kegs spikes @ $4.54	431.30	
	$9,254.25	
Miscellaneous	585.54	
	$9,839.79	
Labor	392.00	

The miscellaneous item covers many second hand tie plates, short rail, material for extra angle bars made on the site, and a portion of temporary construction. The labor item represents handling, unloading and work on angle bars incident to different punching of rails.

Account 7304—Frogs and Switches

This item covers the cost laid in track of the following number of frogs purchased from the Cincinnati Frog & Switch Co.:

9 No. 9 frogs with switches
7 No. 4 frogs with switches
2 No. 2½ frogs with switches.

It also covers labor incident to replacing 4 or 5 old frogs laid previous to the arrival of the new material. Included with the old installation labor is the cost of making several new switch points.

Account 7305—Laying, Surfacing and Ballasting.

This account covers the laying of all tracks in the industrial system about the smelter and the ballasting of such tracks where ballasting was required. The total amount of track laid was 17,150 ft., of which 697 ft. was upon steel trestles.

The ballast used varied according to the conditions prevailing at the time. A large amount of the track was ballasted with material borrowed near the site, while other portions were ballasted with red oxide tailings from the leacher at the cost of $0.25 a yard, either dumped upon the track and spread with the use of the locomotive, or because of the excessive degree of track curvature for the locomotive, dumped and hauled to the place required.

The work of track laying and ballasting extended over many months, being prosecuted as the tracks were required for use, and in some cases as the steel arrived or ballast could be obtained. All tracks were standard gauge.

Trolley System

Account 7306—Poles and Setting.

This account covers the cost of stands, poles and ties to which the poles were attached as well as the labor incident to the erection of this work. There were 102 poles made of 4-in. pipe on an average of 16 ft. long set in a cast-iron stand 16 in. high. The accompanying sketches show four different conditions; double track with two poles used on the far sides of both tracks; double track one pole used between two tracks; single track and one pole and bracket on the under side of the slag track cut floor. (See Figs. 1, 2, 3 and 4.)

Account 7306.1—Brackets and Wiring.

This account covers all the material and labor incident to putting up the trolley line brackets, stringing the wire and insulating the work. The length of the trolley system is 7,824 ft. of which 7,346 ft. is No. 000 H. D. grooved copper wire and 478 ft. of 25 lb. rail used through the slag track cut. The cost here has been figured on the lineal feet of system. Below is a list of the majority of items in the material account.

50 Insulator brackets similar to Ohio Brass Co. No. 1,254
200 Trolley brackets similar to Ohio Brass Co. No. 10,998
Six 15° frogs similar to Ohio Brass Co. No. 10,115
Six 15° frogs similar to Ohio Brass Co. No. 10,016
Three 15° frogs similar to Ohio Brass Co. No. 10,388

The messenger wire was ¼-in. seven-strand extra soft galvanized steel. Extension arms were made of 1½-in. pipe, 9 ft. long. One-quarter inch rod was used for arm brace. The voltage carried for the line is 250 D. C.

Account 7306.2—Rail Bonds.

This account covers cost of all material and labor incident to making the electrical bond throughout the trolley system. Each bond required the drilling of two ⅞-in. holes through the web of the rails. For the most part it was necessary to remove and replace the angle bars. Five hundred twenty-one No. 000 cable rail bonds with ⅞-in. hollow comp. studs, 1 ft. 11 in. center to center—type C. P. 2—A. S. & W. Co. furnished complete with drift pins. (See Fig. 5.)

Account 7306.30—Lighting.

This account covers the cost of all material and labor incident to lighting the trolley line. The electricity here used was taken direct from the trolley lines. Fifty-seven lamps—240 volt—120 watt carbon filament were installed. Four thousand feet of wire were used.

Rolling Stock

Account 7307—Cars, Electric Locomotives, etc.

This account covers the cost of the following equipment together with the labor of unloading, trying out, removing the air brakes from the slag cars which came in on their own wheels, installing extra controllers on slag cars, and a large amount of repair work not carried by account 7307.1.

	Factory Cost	Freight	Total
Three 18-ton calcine cars, weight of each including electrical equipment 33,200 lb. from Kilbourne & Jacobs Mfg. Co.....	$7,200.00	$1,394.22	$8,594.22
Four 225-cu. ft. capacity electrically operated slag cars, from M. H. Treadwell Co. Cars are side dumped by 15-h.p. motors; can also be dumped by hand. Weight of one car 57,000 lb.....	11,620.00	513.48	12,133.48
Two 18-ton electric locomotives from Westinghouse Mfg. Co. Draw-bar pull 8,000 lb. Each locomotive has two 84-h.p. motors. Weight of each locomotive is 43,950 lb.....	8,500.00	1,380.03	9,880.03
One clinker larry car, 12½ tons capacity, 10-h.p. motor, weight 11,000 lb.....	1,450.00	176.00	1,626.00
Two trailer cars, 165 cu. ft. or 25,000 lb. capacity, weight of each car, 5,000 lb...	680.00	160.00	840.00
Two 15-h.p. 220-volt drum type controllers for electric locomotives to dump slag cars	.....	.....	165.64
One pair Schoen rolled steel wheels with axles for slag cars above.....	.....	.....	115.66
Miscellaneous material.....	.....	.....	662.60
			$34,017.63

(See Figs. 122, 123, 124, and 125.)

Account 7307.1—Calcine Car Alteration.

The calcine cars with their pantographs on top were too high to operate successfully beneath the roasters. To overcome this difficulty the tracks beneath were lowered 1 ft. and the cab end of the car was cut down 18 in. in height. The pantograph was then placed on the cab end of the car. Much difficulty was also experienced with the pantographs themselves, especially when the cars were on short curves and taking switches. This account covers all the labor and material incident to overcoming these difficulties.

Trestle Approach to Reverberatory Building

Account 7308—Excavation.

The ground excavated was cemented sand and gravel, overlain with soil permeated with caliche. All the work was done by hand, using picks and shovels. The excavated material was cast to the side of the pier holes, and in some cases it was handled three times.

Account 7308.1—Foundation.

The kind of foundation here represented is the pier type. Sixty piers were cast 5 ft. by 5 ft. by 8 ft. One large abutment was 26 ft. high,

including its footing. The mixture of concrete used was 7 parts of gravel and sand to 1 part of cement, mixed by hand and transported in wheelbarrows, an average of 80 ft. There was no reinforcing, but two anchor bolts, ⅞ in. by 2 ft. long, were placed in every footing. About 70 per cent. of the concrete was formed. The pier tops were finished to a perfect elevation. This is true of all other foundations where they support steel columns.

Account 7308.2—Steel Structure.

This steel trestle was a part of a contract between the Arizona Copper Co. and the Kansas City Structural Steel Co., who furnished erected the major portion of the structural steel about the plant, at a ton price of $72.80 f.o.b. El Paso, and the corrugated iron at $81 f.o.b. Pittsburgh. In every case where a steel structure occurs in this cost sheet, an additional amount of money has been expended for a variety of purposes. A new unit price is therefore obtained which varies with the structure in question. The extra expenses entailed are the freight from Pittsburgh and El Paso, the lumber for nailing strips, air lines and power for riveting and erecting, extra trackage to deliver steel within required distance of erection site according to contract, and many smaller items rightly debited here. The unit figure therefore arrived at gives the actual cost of the steel structure as it stands in every case. There were 163.97 tons of structural steel used here.

Account 7308.3—Woodwork.

This account covers the labor and material for the ties, walkways and railing upon the steel trestle. In the case of the ties they were laid for large part upon 173-ft. radius curve and required dapping to accommodate a ½-in. web projection of steel plate sticking up from each of the two girders which they spanned. The material used was as follows:

18,398 ft. b.m. 8 in. by 8 in. ties o.p.	$652.98
9,248 ft. b.m. 2 in. by 4 in.; 4 in. by 4 in.; and 2 in. by 12 in. 872 lb. ¾-in. round iron	21.37
Nails, bolts and miscellaneous	94.57
	$768.92

150-Ton Track Scales—Receiving Yard

Account 7309—Excavation.

The excavation was in tight sand and gravel. It was done with pick and shovel, handled into cars and hauled 300 ft.

Account 7309.1—Foundation.

This was a job of plain concrete mixed 8 parts sand and gravel to 1 part cement, hauled 2,000 ft. by teams and wheeled in barrows 50 ft.

to place. The concrete was principally walls with a few piers. Eighty per cent. of the exposed surface, other than top and bottom, was formed. A great many ¾-in. bolts were set in the concrete. (See Fig. 7.)

Account 7309.11—Cost and Erection.

This account covers the material and erection of one 50-ft., 150-ton suspension platform track scale with type registering beam, graduated by 10 lb. The platform was arranged for two gauges of dead and live rails—4 ft. 8½ in. and 3 ft. The scale was furnished by Fairbanks, Morse & Co. The material portion of this account is divided up as follows: (See Fig. 7.)

	Cost	Freight	Total
One 50-ft. 150-ton suspension platform track scale	$1,351.00	$486.09	$1,837.09
294 rail clips	45.00		
75 stands and 72 rail blocks	211.85	77.36	362.91
Patterns for blocks and stands	28.70		
Structural steel for track scales			1,089.62
Twenty-five 10 by 12 6 ft. ties and 2 by 12 planks covering			55.83
Bolts, washers, round iron, nails, etc			105.41
			$3,450.86

Account 7309.30—Scale House.

This account covers the cost of material and erection of the scale house. The building in plan is 9 ft. 6 in. by 10 ft. with a shed roof. In front it is 10 ft. 6 in. high, and in the rear 8 ft. The studding is 2 in. by 4 in. and the rafters are 2 in. by 6 in. The siding is corrugated iron and on the roof is 1-in. sheathing and composition roofing paper. There are two windows in both front and rear, and a door in each end. (See Fig. 8.)

Bridge No. 1

Account 7310.1—Foundation.

The concrete covered by this account consisted of two footings about 6 ft. by 6 ft. by 55 ft. with two abutments about 3 ft. by 14 ft. by 55 ft. at base to 14 ft. at top. It was plain concrete, 7 parts sand and gravel and 1 part cement, mixed by hand and by machine and wheeled 45 ft. to place. Ninety per cent. of the vertical and inclined surfaces were formed.

Account 7310.10—Bridge No. 1 Steel Work.

This work was covered by the Kansas City Structural Steel Co.'s contract. (See account 7308.20.) The bridge consisted of two girders—each of three 18-in. by 50-lb. by 20-ft. I-beams, connected with bolts and separators and anchored to the foundations with four 1-in. bolts.

Culvert No. 1

Account 7311—Culvert No. 1 Masonry.

This was a stone culvert 354 ft. long, with inside dimensions 4 ft. by 4 ft. The top was built of old 50-lb. rails at $15 a ton, spaced 8-in. centers and rendered tight with stones set with cement mortar in between rails. There is 20 ft. or more fill over the culvert at various points. The stones laid in cement mortar for the sides and bottom were obtained on the site.

Retaining Walls

Account 7312—Excavation.

This was a long narrow cut through fill, earth, and sand and gravel. It was taken out with picks and shovels and transported 200 ft. with slips.

Account 7312.1—Concrete.

The concrete covered by this account was a wall of gravity section, 8 in. at the top, of various heights and 80 ft. long. The mixture used was 7 parts sand and gravel to 1 part cement, wheeled 50 ft. Half of the yardage was hand mixed and half machine mixed.

Account 7312.20—Masonry.

This wall was built of stone which was handy to the site and was laid in cement mortar. The wall was 124 ft. long, 12 in. to 18 in. thick and from 2 to 5 ft. high.

40-Ton Track Scales on Calcine Track

Account 7313—Excavation.

The excavation here covered was a small rectangular cut through tight, red soil, filled with large stones. It was picked, shoveled and wheeled in barrows 50 ft.

Account 7313.1—Foundation.

The concrete under this account was small walls about 30 ft. by 5 ft. by 22 in. and a 10-in. slab. It was mixed 7 parts sand and gravel to 1 cement by hand and wheeled 25 ft. to place. About twenty-six $\frac{3}{4}$-in. bolts were set in the concrete. Eighty-five per cent. of the vertical surface of this concrete was formed. (See Fig. 9.)

Account 7313.20—Cost and Erection.

This account covers the cost of the material and its erection of one 40-ton track scale. The scale platform is 24 ft. long with one 4-ft. 8½-in. track passing over it. It has a type registering beam. The scales were furnished by Fairbanks, Morse & Co. Itemized, the material account stands as follows:

	Cost	Freight	Total
1 40-ton track scale complete	$400	$116.86	$516.86
6-in. by 6-in. ties; 8-in. by 12-in. stringers. 2-in by 12-in. decking			26.19
6-in. and 5-in. channels and 5-in. I-beams			102.85
Strap steel, bolts, pipe, hauling, etc			64.95
			$710.85

(See Fig. 9.)

Account 7313.30—Scale House.

Same as 7309.30—practically.

Trestles to Receiving Bins

Account 7314—Excavating.

This excavation covered two large abutments 6 ft. in the ground and 10 piers going about 15 ft. into the ground. The material excavated was earth and adobe. It was handled in the pier footings by a windlass.

Account 7314.1—Foundation.

The concrete here covered was in two large abutments and in 10 piers. The abutments were about 8 ft. by 16 ft. by 24 ft. and the piers 7 ft. by 7 ft. by 23 ft. Forty-eight $\frac{3}{4}$-in. bolts were set in the concrete. Twenty per cent. of the vertical surfaces were formed. The concrete was mixed in proportions of 7 parts sand and gravel to 1 part cement. A large amount of boulders was used in the piers. The concrete was both hand and machine mixed, and was wheeled in barrows an average of 200 ft.

Account 7314.2—Steel Structure.

There were 109.53 tons of structural steel used here.

Account 7314.30—Woodwork.

The woodwork here was practically the same in every respect as 7308.30, with additional walkways of 2 by 12 planks, nailed to strips bolted to the steel.

16,920 ft. b.m. 8 by 8 ties and 6 by 6 guard rails.

10,286 ft. b.m. 2 by 12, 2 by 4, 4 by 4 walk and railings; 27,206 ft. total b.m. was used.

(See Fig. 6.)

Receiving Bins

Account 7401—Excavation.

This work covered the digging of a number of piers 7 ft. by 7 ft. to a depth ranging from 16 ft. to 23 ft. into gravel. The dirt was easily dug but had to be handled from the lower half of the holes with windlasses. It was carted away at the top 225 ft.

Account 7402—Foundation.

Only 5 per cent. of this concrete was formed. The lower part was machine mixed in proportions of 12 gravel and sand to 1 cement, while the upper part was 6 to 1. It was wheeled 200 ft. to place. The top surfaces were trowel finished to a perfect elevation for receiving the steel.

Account 7403—Steel Structure.

There were 11.35 tons of corrugated and 341.74 tons of structural steel used here. (See account 7308.2. See Fig. 11.)

Account 7404—Gates.

This account covers the cost of material, unloading, hauling, fabrication, alteration, and erection of 30 gates beneath the receiving bins. All cast-iron parts together with operating wheel, shaft and gate itself were purchased outright.

The chutes attached to the gates were fabricated in the new smelter shops. The parts were assembled in the field and there erected. The holes in the steel structure to which the gates were attached had to be rebored in the field. The counterweights for the 12 coarse ore-bin gates were made on the job and erected. These 12 counterweighted gates are opened by rack and pinion, operated by a hand wheel and cut up through the stream. The chutes to guide the ore to the feeder are of $\frac{3}{8}$-in. plate, while the gate is $\frac{5}{8}$ in. The other 16 gates for the concentrate bin are similar to the above, but are not counterweighted and cut down through the stream. (See Fig. 12.)

Account 7405—Conveyor No. 1.

This account covers the entire labor and material connected with the installation of conveyor No. 1. It does not include the steel frame to which the idlers are attached, but does cover the cost and installation of the traveling feeder with the necessary ties, rails, wire and motor. This segregation of charges is true of all conveyor costs given in this cost sheet. All the conveyors were furnished by the Robins Conveying Belt Co. Conveyor No. 1 was a 30-in. belt, making a 97-ft. conveyor with a 3-ft. rise, operating at a speed of 150 ft. per minute, capable of handling 100 tons per hour. It is supplied with 12-in. material from the bins above it through a speeded feeder. (See Fig. 6.) The material account is segregated as follows:

	Cost	Freight	Total
202-ft. 30-in. belt	$686.80	$46.45	
Feeder belt	56.75		
Conveying idlers, etc.	1,497.12	185.09	
Cent. switch	34.00	2.20	
Broken pulley	44.00		
Miscellaneous material	307.73		
One 5-h.p. motor	87.05		
	$2,713.45	$233.74	$2,947.19

(See Fig. 13.)

Account 7405.01—Conveyor No. 2.

This conveyor is similar to No. 1. It has a 20-in. belt, making a conveyor 117 ft. long, with a 3-ft. rise, operating at a speed of 200 feet per minute with a capacity of 150 tons and taking $\frac{3}{8}$-in. concentrates through a speeded feeder. The material account is segregated as follows:

	Cost	Freight	Total
241 ft. 6-in. to 20-in. belt	$432.38	$31.94	.....
Feeder belt	56.75	.....	.....
Robins material	1,367.50	169.07	.....
Cent. switch	34.00	2.21	.....
Miscellaneous material	317.14	.....	.....
5-h.p. motor	87.04	.....	.....
	$2,294.81	$203.22	$2,498.03

(See Fig. 13.)

Account 7407—Lighting.

The receiving bins were lighted with 22 drop lights using 100 volts A. C. current.

CRUSHING PLANT

Account 7701—Excavation.

This was a large rectangular cut for the crusher building made through cemented sand and gravel, with streaks of soil running through the cut hardened by caliche. The work was done with pick and shovel and handled by wheelbarrow into carts and hauled 225 ft.

Account 7702—Foundation.

This concrete was reinforced with $\frac{1}{2}$-in. and $\frac{3}{4}$-in. round, medium steel rods. It was cast in walls, 12 in. thick, about 12 ft. high and as a 12-in. floor slab. The concrete was machine mixed in the proportions of 5 sand and gravel to 1 cement and about 60 per cent. of the vertical surfaces were formed. It was wheeled in barrows 400 ft. up an 8 per cent. grade.

Account 7703—Steel Structure.

There were 5.17 tons of corrugated iron and 19.90 tons of structural steel used. (See account 7308.2.)

Account 7703.1—Doors, Windows and Frames.

The material here used for openings was as follows:

9 windows and frames 3 ft. $10\frac{5}{8}$ in. by 7 ft. 8 in., 24 lights. 4 sash and frames, 2 ft. $11\frac{1}{4}$ in. by 3 ft. $11\frac{3}{4}$ in., 9 lights.	$138.40
Balances, butts, catches, etc	25.38
Lumber for sills	6.93
	$170.71

Account 7703.2—Painting Woodwork.

All the woodwork was painted with two coats of lead and linseed oil in cream color.

Account 7704—Crushing Machinery.

This account covers the material cost and labor of installing the following machinery:

One 36-in. by 18-in. Farrell crusher, second hand, weight 50,000 lb	$1,000.00
Miscellaneous lumber	93.61
	$1,093.61

Account 7704.1—Chutes.

This account covers the cost of material noted below, the labor of fabricating the spouts, hoppers, and their erection:

1 Grizzly screen 3 ft. by 8 ft. made of 1-in. bars, having 2½-in. by 2½-in. openings, framework made of $\frac{5}{16}$-in. plate and 3-in. by 3-in. angles.

1 Spout for grizzly, dumping on conveyor No. 4 made of $\frac{5}{16}$-in. steel plate and 2-in. by 2-in. by ¼-in. angles. ¾-in. C.I. liners used.

1 Hopper for crusher made of $\frac{5}{16}$-in. steel plates 2½-in. by 2½-in. by ¼-in. angles. ¾-in. C.I. liners used.

Account 7705—Shafting, Pulleys and Belting.

This account covers the material cost and erection of the following:

1 Pc. shafting 4 $\frac{7}{16}$ in. by 6 ft. 6 in. with two collars	$29.64
1 Pc. shafting 4 $\frac{7}{16}$ by 5 ft. 3 in	16.51
Two 4 $\frac{7}{16}$-in. rigid pillow blocks	58.55
One 48-in. by 11-in. split pulley	47.59
One 36-in. by 16-in. split pulley	43.71
Two 4 $\frac{7}{16}$-in. safety collars	5.39
43 ft. 10-in. two-ply leather belt	60.80
42 ft. 14-in. two-ply leather belt	220.37
Miscellaneous	0.79
	$483.35

Account 7706—Motor.

This account covers the cost of the following material and the installation:

One 50-h.p. Crocker-Wheeler squirrel-cage motor with starter	$478.41
Lumber for housing motor	35.23
	$513.64

Account 7707—Power Wiring.

Account 7707.1—Lighting.

This account covers the cost and installation of the following material:

8 drops 16 candle power
30 ft. brewery cord
350 ft. No. 12 weatherproof wire
45 ft. 1-in. conduit.
Switches, bolts, etc.

SAMPLING PLANT

Account 7801—Excavation.

This account covers the excavation of the sampling plant and the necessary backfill tamped in 5-in. layers in the low parts where the basement concrete floor was cast. It was done with picks, shovels and wheelbarrows, through earth, sand and gravel.

Account 7802—Foundation.

This concrete was cast in the walls and piers of the sampling plant. It was mixed by machine in the proportions of 7 parts sand and gravel to 1 part cement and wheeled in barrows 150 ft. Ninety per cent. of the vertical surfaces of the concrete was formed. The cost of all anchor bolts as well as the finish to exact level for building columns is included here.

Account 7802.1—Concrete Ground Floor.

This was plain concrete floor laid with sand joints in about 6-ft. square blocks 5 in. thick, in the proportions of 5 parts sand and gravel to 1 cement. The top finish was 1 in. thick, made 2 parts sand to 1 cement, troweled smooth. The concrete was mixed by machine and transported 175 ft. in barrows.

Account 7802.2—Reinforced Concrete Floors.

This concrete was cast over steel I-beams, using forms between the steel beams. The mix was the same as the above floor with the same top finish. The reinforcing used was Clinton welded fabric 2-in. by 12-in. mesh. The floor was 4½ in. thick. The concrete was mixed by machine and transported 500 ft. to place by wagon, wheelbarrow and hoist.

Account 7803—Steel Structure.

There were 13.46 tons of corrugated iron and 97.39 tons of structural steel used. (See account 7308.2.)

Account 7803.1—Doors, Windows and Frames.

This account covers the purchase price and erection cost of material enumerated below. The doors are not given in the list, as they were made upon the job, but correspond to the frames noted. Necessary hardware is also included in the cost.

Fourteen 24-light windows 3 ft. 9½ in. by 7 ft. 8 in. by 1¾ in. with frames
Twenty-nine 40-light windows 7 ft. 5¾ in. by 3 ft. 10¾ in. by 1¾-in. with frames

Two 9-light windows 2 ft. 11¼ in. by 3 ft. 11¼ in. by 1¾ in. with frames	
One 4-ft. 8-in. by 7-ft. 4-in. door frame	
One 9-ft. 2-in. by 8-ft. 10-in. door frame	
Three 4-ft. 8-in. by 7-ft. 4-in. door frames	
Five 3-ft. 8-in. by 7-ft. 4-in. door frames	
One 3-ft. 8-in. by 7-ft. 2-in. door frame	
One 9-ft. 2-in. by 9-ft. 2¾-in. door frame	$564.90

(See Fig. 14.)

Account 7803.11—Painting Doors and Windows.

This covers the material and labor of applying two coats of linseed oil and white lead.

Account 7804—Shafting, Pulleys and Belting.

This is not a good cost. The labor is unquestionably too low and has been absorbed by some of the following accounts up to account 7807.5. Eighty-five feet of shafting, varying in size from $1\frac{15}{16}$ in. to $3\frac{7}{16}$ in. with 28 pulleys of various diameters and face, with the various hangers, collars, etc., and 1,325 ft. of 3-in. to 12-in. leather belting were to be taken care of here. The material is correct. (See Fig. 108.)

Account 7805—Motors.

The material covered by this account is as follows:

One 15-h.p. squirrel-cage motor	$189.94
One 75-h.p. slip ring	644.00
Overload release	14.25
Miscellaneous	39.38
	$887.57

Account 7806—Power Wiring.

Account 7806.1—Lighting.

36 drop lights	
No. 12 weatherproof wire used in conduit	$140.57

Account 7807—Rolls and Samplers, Cost and Erection.

This cost covers the price and installation of the following material, together with the application of two coats of Dixon's silica graphite paint upon the rolls, samplers and chutes:

	Cost	Freight	Total
2 sets 24 by 12 rolls, from Chalmers & Williams	1,330.00	344.16	1,674.16
1 set 42 by 16 rolls, from Chalmers & Williams	1,567.50	482.40	2,049.90
1 set 48 by 12 rolls, from Chalmers & Williams	1,710.00	716.43	2,426.43
One 27-in. Snyder sampler with 28-in. by 3½-in. pulley T. & L.	38.00	5.04	43.04

	Cost	Freight	Total
Two 42-in. Snyder samplers with 40-in. by 3½-in. pulley T. & L.....	133.00	29.95	162.95
1 No. 1 Vezin sampler with spout arranged for 5 per cent. cut, Allis-Chalmers.....	162.00	20.51	182.51
1 No. 3 Vezin sampler with spout arranged for 10 per cent. cut, Allis-Chalmers.....	220.00	45.64	265.64
Two 5-T steel plate crawls for 12-in. I's...	104.00	15.76	119.76
One 4-T steel plate crawl for 10-in. I.....	40.00	6.11	46.11
Two 3-T steel plate crawls for 9-in. I's....	64.00	8.61	72.61
One 5-T Triplex chain block.....	112.00	14.49	126.49
One 4-T Triplex chain block.....	88.00	9.10	97.10
Structural steel, machine parts.....	.....	.....	632.44
Miscellaneous, etc.....			$7,899.14

(See Figs. 15, 16, 110, 111 and 112.)

Account 7807.10—Cast Iron Liners and Drying Pan, and Erection of Chutes.

This account covers the erection of account 7807.5, and the cost and erection of the following:

1 Sample drying pan 8 ft. ½ in. by 5 ft. 2½ in.....	$136.11
½-in. C.I. liners for chutes, castings.....	865.75
	$1,001.86

Account 7807.20—Elevators.

Below is a description of the elevator with the material cost. The erection is too low and not usable.

33 ft. between head and tail shafts
Head and tail pulley 34 in. by 9 in.
Drive pulley 23 in. by 6 in., 40 h.p.m.
All housed with No. 10 plate steel casing
72 ft. 6 in. of 8-in. 6-ply rubber belt
Forty-eight 6-in. by 4-in. malleable buckets, style A.A. Mfrs. std.
Liners for casing made of white iron ¾ in. and 1 in. thick
Cost..... $458.22
(See Fig. 11.)

Account 7807.5—Steel Chutes.

This is the average cost per pound of material and labor for fabricating all steel chutes used in the sample mill. In general they were made of $\frac{3}{16}$-in. plate and light angles. The cost of erecting is in 7807.1 and in the comparative costs are found the individual chute costs. (See Fig. 109.)

Account 7809—Keystone Plate Partitions.

1,523 sq. ft. of No. 14 Keystone Plate partitions were erected by riveting the plates together and attaching them to the structural steel of the building. The plates were orginally intended for a roof upon the roaster dust chamber and had each long edge turned up 1½ in. The cost of cutting these edges off is here included. (See Fig. 113.)

Account 7810—Alteration of Chutes and Machinery.

BEDDING PLANT AND BUNKER BINS

Account 7901—Excavation.

This excavation involved making long, deep, oblong cuts through earth and sand and gravel bonded with caliche. It was necessary to use powder to shake up the ground, followed in some cases with plows. A part of the work was handled with slips and fresnos; another part by picks, shovels and wagons. The average haul was 600 ft.

Account 7902—Foundation.

This concrete yardage was made up as follows:

550 ft. footing, 4 ft. wide by 2 ft. deep—plain
550 ft. wall, 1 ft. 6 in. thick by 5 ft. high—plain
1,200 ft. footing, 3 ft. 6 in. wide by 10 in. deep—plain
600 ft. wall, 1 ft. 6 in. thick by 6 ft. high—plain reinforced, coping
600 ft. wall, 1 ft. 6 in. thick by 11 ft. high—plain
320 ft. footing, 7 ft. wide by 3 ft. deep, reinforced with ¾-in. and ½-in. rods, 6 in. on centers
320 ft. wall, 4 ft. 5 in. thick by 11 ft. high, reinforced with ¾-in. rods, 12 in. on centers
12,700 sq. ft. rough finished slab, 5 in. thick

The concrete, of which 85 per cent. of the vertical surface was formed, was machine mixed, in the proportions of 7 parts sand and gravel to 1 part cement. It was wheeled to place in barrows, a distance on the average of 180 ft. About half of the yardage was reinforced.

Account 7903—Steel Structure.

(See account 7308.2.)

Structural steel	510.41	tons
Corrugated iron	38.30	tons
	548.71	tons

(See Fig. 114.)

Account 7904.3—Bunker Bin Gates.

These are cast-iron chutes about 22 in. square with an arc gate controlling the discharge through the bottom. All cast iron is $\frac{3}{4}$ in. thick, save the $\frac{5}{8}$-in. wearing plate upon the arc gate. The operating lever was furnished in the structural steel contract.

42 spouts $\frac{3}{4}$-in. cast iron
42 gates $\frac{3}{4}$-in. cast iron
42 cover plates $\frac{5}{8}$-in. cast iron $1,021.64

Account 7904.4—Chutes for Conveyors 7[1] to 10[2] inc.

This account covers the fabrication, erecting and material in the chutes directing the ore from one belt to another at the beds and bunker bins. Also included here are the cast-iron wearing plates for lining the chutes.

The chutes are made of $\frac{3}{16}$-in. plate and the necessary angles. The wearing plates are $\frac{3}{4}$ in. to 1 in. thick, hard, white cast iron.

Account 7905—Two Reclaimers.

(See Fig. 116.)

Account 7905.1—Two Reclaimers Wiring.

This account covers the material cost and labor of installing a double trolley wire on three beds together with the wiring of two reclaimers and switchboards. The material is wire, condulets, circuit breakers, and the like. (See Fig. 118.)

Account 7906—Lighting.

This account covers the labor and material used in lighting the bedding plant and bunker bins.

63 carbon lamps, 16 candle power
455 ft. brewery cord
1,395 ft. No. 8 weatherproof wire
1,835 ft. No. 12 weatherproof wire
205 ft. 1¼-in. conduit
285 ft. 1-in. conduit.

Account 7907—Transfer Car.

This car was a structural steel frame about 4 ft. high and 62 ft. by 18 ft. in plan. It transfers the reclaimers from one bed to another or to the repair shed under its own power, using a 7½-h.p. motor, getting direct current through a trolley. It was furnished by the Robins Conveying Belt Co. (See Fig. 117.)

Account 7908—Signal System.

This account is of no value.

ROASTING PLANT

Account 8101—Excavation.

This covered a large surface grade to the required elevation of the site, made by plows and fresnos and hauling the dirt to a railroad grade an average of 450 ft. This was followed by picks, shovels and carts, making deep cuts to gravel through red clay and boulders for the steel foundation.

Account 8102—Foundations.

This concrete was all cast as piers with at least 2 ¾-in. bolts in each pier. Only 10 per cent. of the vertical surfaces was formed, though many of the piers were 6 ft. deep. One-half of the concrete was machine mixed and one-half was hand mixed in the proportions of 7 parts sand and gravel to 1 cement. The pier tops were finished to a perfect elevation to receive the steel columns. The concrete was wheeled about 60 ft.

Account 8103—Steel Structure.

(See account 7308.2.)

There were 23.16 tons of corrugated iron and 422.12 tons of structural steel used. (See Figs. 117 and 118.)

Account 8103.1—Elevator.

This account gives the entire labor and material incident to erecting a 52-ft. high 10-ft. 6-in. by 8-ft. 6-in. platform elevator in a self-supporting structural steel frame. It does not include the excavation and concrete pit. The account segregated stands thus:

Structural steel erected for elevator frame and tower	$840.97
One 5-T electric hoist with 240-volt D.C. motor	1,261.60
3,792 b.f. lumber	11.56
Rope sheaves, counterweights, etc	29.23
Labor	37.79
Miscellaneous	8.47
	$2,189.62

The labor installed the motor, hoist and wood platform.

Account 8104—Roasters, Cost and Erection.

This account covers the cost and erection of the roaster shells as furnished by the Kansas City Structural Steel Co. and the roaster equipment, namely, the central shaft, rabble arms, rabbles, driving mechanism, doors to roasters, cast-iron rings, etc. As segregated the material account shows below. The Herreshoff furnaces have 6 superimposed hearths and a top drying hearth. The arms are cooled by air furnished by two motor-driven fans. The diameter of the shell is 21 ft. 7½ in. outside and 18 ft. 2 in. in height.

	Cost	Freight	Total
8 furnaces from Pacific Foundry Co. designed by Gen. Chemical Co.	$36,278.12	$8,016.73	$44,294.85
Eight 1-ton steel trolleys for 6-in. I's (See Fig. 121.)	.....	.....	155.30
One 1-ton duplex chain block	.....	.....	27.47
Steel shells erected—KCSSCo	10,192.73	840.06	11,032.79
Power for riveting			452.25
Miscellaneous			363.43
			$56,326.09

Account 8104.1—Roaster Alteration.

Account 8105—Brickwork.

This account covers the cost of the brick, mortar materials, etc., and labor of the mason with helpers of installing the brick in the roasters. The unloading of the brick from cars, the centers and carpenter labor are taken care of elsewhere. The brick here used were in the main special shapes and 95 per cent. of them were hard burned red brick. In all there were 15 different shapes. The hearths were laid dry and the shell brick laid with slimes from the copper company's concentrator. (See Fig. 133.)

Account 8105.01—Brick Unloading.

This covers the cost of leveling ground, checking up quantities, and unloading of brick from cars to roasters. This total cost includes the unloading of all brick for roaster use, the actual amount needed plus the extra not used.

Account 8105.02—Brickwork Centering.

This account covers the cost of making, installing, removing and the material for 16 sets of centers used for putting in 48 hearths. (See Fig. 133.)

Account 8106.01—Roaster Flue Spouts.

This covers the cost of material and erection of 10 spouts with gates from the roasters' common flue. The material is as follows:

10 cast-iron gates for 12¼ in. diameter spouts.

10 spouts of No. 10 plate, 24 ft. long, 12¼-in. outside diameter, fastened to base of hopper by 2 in. by 2 in. angle collar.

10 ⅜-in. plate slides for gates.

(See Fig. 135.)

Account 8106.02—Tile Work.

This account covers the mason labor, carpenter labor, cost of tile and unloading, mortar materials and a lumber charge for scaffolds used in this flue. The flue is built of tile and is about 50 ft. from the ground. The tile for the roof was laid between T iron spanners from wall to wall. The mortar was lime. (See Fig. 132.)

Account 8106.03—Painting Flue.

The inside of the above flue was given one coat of silicate of soda. The account covers this material and labor cost.

Account 8107—Shafting, Pulleys and Belting.

Below is the material list erected under this account. The shafting was attached directly to the steel frame of the building.

47 ft. of $3\frac{3}{16}$-in. shafting 103 ft. of $2\frac{5}{16}$-in. shafting Two 20-in. by 12-in. pulleys One 36-in. by 10-in. with clutches for each roaster	.....	$1,500.50
150 ft. 10-in. 6-ply rubber belt 310 ft. 8-in. 7-ply rubber belt	.....	471.85
Miscellaneous		27.54
		$1,999.89

Account 8108—Motor.

This covers the cost of material and labor of installing one 30-h.p. motor to drive the roasters. It is located directly upon the first steel floor of the roaster building.

One 30-h.p. squirrel-cage motor	$267.80
One overload release	14.25
Miscellaneous wire, insulators, belt, etc.	181.91
	$463.96

Account 8109—Lighting.

The roasters are furnished with light on all floors.

Account 8112—Motor-driven Fans.

This covers the price and cost of installing upon their foundations 2 motor-driven fans, which furnish the air to cool the roaster arms. They are 55-in. double width, full housing conoidal fans, direct connected, each with a 25-h.p. squirrel-cage induction motor.

Each fan has a capacity of 22,000 cu. ft. of air per minute against a pressure of $1\frac{3}{4}$ in. water.

	Cost	Freight	Total
2 fans and motors	$1,203.00	$199.49	$1,402.49
Miscellaneous			3.42
			$1,405.91

Account 8112.1—Blast Pipe.

This account covers the material price, cost of fabrication and installation of 240 ft. of blast pipe. The installation referred to is connecting up and riveting the pipe in place in the field only. The pipe is made of No. 10 and No. 12 plate and varied in diameter from 18 in. to 36 in. The inlet pipe to each roaster was 18-in. diameter.

Account 8113—Conveyor No. 12.

(See account 7405.) This conveyor takes the material of the beds from conveyor No. 11 and delivers it to conveyor 13^1 and 13^2. It is a 20-in. belt, 51 ft. 3 in. from center line of tail pulley to center line of head pulley, with an 8-ft. rise, operating at a speed of 300 ft. per minute, with a capacity of 100 tons per hour. The segregated material account is as follows:

	Cost	Freight	Total
Belt	$209.43	$14.43	$223.86
Conveyor material	370.91	45.86	416.77
One 5-h.p.m.	.....	.....	87.04
1 centrifugal switch	34.00	2.21	36.21
Lumber, decking and painting material			30.97
Spout conv. 11 to conv. 12			7.70
Miscellaneous			2.50
			$805.05

Account 8113.1—Conveyors 13^1 and 13^2.

Conveyors 13^1 and 13^2 take the product from conveyor 12 running the length of the roaster building; each delivers the material through a separate automatic tripper to the roaster bins. They are identical. Both are 20-in. belts, running perfectly flat, 109 ft. from center line of head pulley to center line of tail pulley, operating at a speed of 300 ft. per minute, with a capacity of 100 tons per hour. The account for material stands as follows:

	Cost	Freight	Total
Belt	$837.72	$59.26	$896.98
Conveyor material	913.60	236.45	2,150.05
2 automatic trippers & track	1,000.00		
Two 5-h.p. motors	.....	.....	174.08
2 centrifugal switches	68.00	4.42	72.42
Drive belt			30.79
Lumber, decking, paint			123.90
Spouts from conveyor 12			11.54
Miscellaneous			12.57
			$3,472.33

(See Fig. 91.)

Account 8113.2—Stile over Conveyors 13^1 and 13^2.

These stiles were made of structural steel, purchased from the Kansas City Structural Steel Co. and erected by the Arizona Copper Co.

ROASTER DUST CHAMBER

Account 8121—Excavation.

Same as 8101.

Account 8122—Foundation.

Same as 8102.

Account 8123—Steel Structure.

(See account 7308.2.)

There were 14.8 tons of No. 14 Keystone plate, 376.4 tons of structural steel used here and 27.63 tons of No. 11 Keystone plate. (See Figs. 103 and 131.)

Account 8123.01—Wire Baffles.

This account covers the cost of material, labor, and repairs entailed in installing 60,480 wire baffles in the roaster dust chamber. The wires with hooks on one end like shepherds' crooks were hung 4 in. on centers both ways from chains supported from the lower members of the roof trusses 4 in. apart. The segregation of material is as follows. The wires hung a few inches off the dust chamber bottom and were thus of various lengths. (See Fig. 131.)

1,008 ⅜-in. chains 20 ft. 6 in. long with two hooks	$1,451.00
63,964 lb. No. 10 wire (black)	2,877.82
6,557 lb. No. 10 wire (black)	361.96
Miscellaneous	67.35
	$4,758.23

(See Fig. 131.)

Account 8123.1—Tile Work.

The sides of the roaster dust chamber and inclined bottom were built of 4-in. hollow tile. This material, labor of masons and their helpers, lumber for scaffolds, carpenter labor, mortar, material and power for hoisting are here included in the cost.

Account 8123.11—Tile Unloading.

This account covers the cost of unloading, wheeling, checking quantities and leveling up ground to receive tile.

Account 8123.2—Painting Outside.

The outside of the tile portion of the chamber, namely, sides and bottom, was given one coat of mineral red and linseed oil. The mortar was scraped from the tile before applying. This account covers the labor and material.

Account 8123.3—Painting Inside.

The tile work on the inside of the dust chamber was given one coat of silicate of soda used as a paint. This account covers the labor and material of this operation.

REVERBERATORY PLANT

Account 8301—Excavation.

This was the making of a deep surface cut for the building. The material was principally red clay and boulders. In many cases powder was used. In general the ground was plowed, scraped with fresnos through a trap into narrow-gauge side-dump cars and hauled 2,000 ft. by steam locomotive to make a railroad fill.

Account 8301.01—Backfilling.

This covers the cost of backfilling in the reverberatory bottoms and between the reverberatories. The dirt was red clay soil. It was plowed, hauled in wagons, dumped, shoveled into a derrick box and lifted by a locomotive crane over the reverberatory sites and dumped. It was then distributed with wheelbarrows and tamped in 4-in. layers.

Account 8302—Foundation.

This work consisted of long walls averaging 230 ft. long, 4 ft. at top and 6 ft. at bottom, and of beams to withstand the reverberatory buck stay pressure, averaging 660 ft. long by 3 ft. by 3 ft. The walls were reinforced with $\frac{3}{4}$-in. rods, spaced about 4 to 6 in. on centers, while the beams were reinforced with $\frac{3}{4}$-in. and 1-in. rods about 4 in. center lines one way.

The mixture used was 1 part cement and 5 parts sand and gravel, machine mixed, transported 100 ft. average, with wagons, cars, wheelbarrows and concrete carts as the situation demanded. 100 per cent. of the vertical surface was formed.

Account 8302.1—Concrete Counterweights.

These are used for the cross and header flues. Some are 1 ft. square from 6 ft. to 10 ft. long, cast in wooden forms. Others are circular, cast in steel cylinders. The concrete was mixed in a machine, wheeled 150 ft. and made plain with 1 part cement to 5 sand and gravel.

Account 8303—Steel Structure.

There is in this building 55.31 tons of corrugated iron, and 405.78 tons of structural steel.

Account 8304.—Reverberatories—Brickwork.

This account covers all the brick, mortar material and mason labor used in laying the brick of three reverberatories. The overall dimensions of the furnaces are 104 ft. long by 27 ft. wide and about 10 ft. 6 in. to the crown of the arch. The side walls are 2 ft. 6 in. thick and the arch is 20 in. deep. In the three reverberatories there were used 106,350 red brick laid in lime mortar and 257,288 various shaped silica brick dipped in silica slimes. (See Figs. 77, 78 and 79.)

Account 8304.01—Unloading Brick.

This account covers the cost of preparing the unloading site, building three brick sheds of 84,000 cu. ft. capacity, the unloading of the brick, checking the quantities, and piling separately 31 different shapes.

Account 8304.02—Centering.

This account covers the cost of material, fabrication of one center together with the labor and erecting and tearing it down three times. The arch was made of 2 by 12 centers with 1-in. sheathing tacked on top. The 2 by 12 centers were spaced 18 in. centers, supported on 6 by 8 stringers held up by 4 by 8 posts suitably braced. (See Fig. 130.)

Account 8304.05—Rehandling Brick.

This account covers the transporting at many different times of the silica brick from the sheds to the reverberatories a distance of 500 ft. by one-mule carts.

Account 8304.1—Steel Work.

This account covers the cost of the material and labor of installing the steel buck stays for the reverberatories. Below is a list of material:

Buck stays	276 12-in. 31.5-lb. beams on sides
Buck stays	84 12-in. 31.5-lb. beams on ends
Rails	1,212-ft. 60-lb. rails
Rails	594-ft. 75-lb. rails
Cross stay rds.	78 1¾ in. diam. 31 ft. 8 in. long
Longitudinal rds.	27 1¼ in. diam. 110 ft.

6 steel supports for longitudinal rods made of 2 to 8-in. angles 11½ lb.
(See Fig. 78.)

Account 8304.2—Silica Fill.

This cost is for the silica purchased, crushed in a variety of ways, transported to the furnaces and tamped in place there in layers. It came from the Calumet & Arizona Mining Co., at Douglas. The segregation of the account per ton is as follows:

First cost	Freight	Crushing and placing	Total
$2.757	$1.749	$3.134	$7.64

Account 8304.3—Hoppers and Chutes.

This account covers the cost of material and the installation of feed hoppers and chutes with their gates and levers to the reverberatories.

Cast-iron hoppers, chutes, weights, levers, bars	$1,216.20
300 ft. ½-in. steel sash cord, 100 clips, 50 thimbles	23.00
102 lb. ¼ by 3 flat iron	2.45
200 lb. 1 5/8-in. shafting	12.09
Miscellaneous	9.85
	$1,263.59

(See Figs. 81 and 82.)

Account 8305—Cross and Header Flues—Brickwork.
This covers the cost of the tile, brick, mortar, lumber for scaffolds, mason and carpenter labor incident to building the cross and header flues from the reverberatories to the boilers. They are 11 ft. from the ground and 8 ft. 6 in. by 8 ft. 6 in. in section. The roof is a brick arch held by buck stays. There were red brick, fire brick and 4-in. bottom tile used in the construction. (See Figs. 80, 83, 84 and 85.)

Account 8305.1—Cross and Header Flues—Unloading Brick.
This covers the unloading, checking quantities and preparing site for the brick used in the flues.

Account 8305.02—Cross and Header Flues—Centering.
This covers the cost of material and labor for making, installing and wrecking the arch center for the flues of 8305.01. Centers were made for about one-half the length and then moved to the other half.

Account 8305.2—Cross and Header Flues—Painting Brick.
This covers the cost of painting the outside of the flues with one coat of mineral red and linseed oil when the flues were in service. The steel work of the buck stays was given at the same time one coat of graphite paint.

Account 8306—Flues, Boilers to Reverb. Flue—Excavation.
This work covers small pier excavation in red clay. It was picked, shoveled into barrows and transported about 15 ft.

Account 8306.1—Flues, Boilers to Reverb. Flue—Foundation.
This foundation is some small piers of plain concrete mixed by machine, 1 part cement to 6 parts sand and gravel, transported by wheelbarrows 125 ft. About 40 per cent. of the vertical surfaces was formed. Every pier has 2 $\frac{5}{8}$-in. anchor bolts.

Account 8306.2—Flues, Boilers to Reverb. Flue—Steel Structure.
(See account 7308.2.)
This covers seven 6-ft. diameter flues of $\frac{1}{4}$-in. steel with their supports. There were 34.78 tons.

Account 8307—Boiler Building—Excavation.
This account covers the digging of two long deep cuts for retaining walls. Two feet of clay were encountered, followed by sand and gravel and boulders with caliche. The ground was partly blasted, all picked, shoveled into wagons and hauled a distance of 600 ft.

Account 8307.01 Waste Heat Boilers—Excavation.
This work was digging shallow trenches for small foundations, through red clay and small boulders. The ground was picked, shoveled and hauled 600 ft.

Account 8307.02—Oil-fired Boilers—Excavation.
Same as 8307.01.

Account 8307.03—Boiler Feed Pumps—Excavation.
This was a deep square cut involving 659 cu. yd. through red clay and boulders, into sand, gravel and boulders tightened with caliche. It was partly loosened with powder, picked, shoveled and hauled by wagons 400 ft. The lower half was handled twice, once onto scaffolds and the second time out of the pit.

Account 8307.04—Backfill, Back of Boiler Wall.
This was filling behind a long retaining wall. This dirt was adobe, wetted and tamped in 5-in. layers. The dirt was wheeled 60 ft. to place.

Account 8307.1—Boiler Building—Foundations.
This work covered a reinforced wall 240 ft. long, 11 ft. high, 1 ft. at top, 2 ft. at bottom, with pilasters connected at top with horizontal reinforced concrete beams forming the support for waste heat and oil-fired boilers. Three-fourth inch and ¾-in. rods spaced about 6 in. to 8 in. were used together with many foundation bolts. The mixture was 5 sand and gravel to 1 cement, made in a machine and hauled 175 ft. in wagons, thence by wheelbarrow 10 to 15 ft. into place. One hundred per cent. of the vertical surfaces was formed.

Account 8307.11—Waste Heat Boilers—Foundations.
This work covered the reinforced concrete beams noted in 8307.1 required for the waste heat boilers. The other conditions were the same, save 50 per cent. only of the vertical surfaces was formed.

Account 8307.12—Oil-fired Boilers—Foundations.
(See account 8307.11.)

Account 8307.13—Feed Pumps—Foundation.
This account covered the reinforced cantilever walls for a pit 26 ft. by 36 ft. in plan, 14 ft. high and 1 ft. thick. The rods were ½ in. and ¾ in. The mixture was machine mixed 5 sand and gravel to 1 cement, hauled 175 ft. in wagons to place. Seventy-five per cent. of the vertical surfaces was formed.

Account 8307.2—Floor over Slag Track Cut—Floor.
This covered the laying of a 6-in. reinforced concrete floor 30 ft. by 240 ft. over steel I-beams with a mortar finish troweled smooth. The mix was machine made 5 sand and gravel to 1 cement, with 2 to 1 top finish. A $\frac{3}{16}$-in. woven wire triangular mesh was used and 50 per cent. of the surface was formed. The concrete was wheeled in barrows an average of 175 ft. (See Fig. 86.)

Account 8307.3—Floor around Boilers.

This is a 4,000-sq. ft. plain concrete floor of 4 in. laid in blocks with sand joints and given a 2 to 1 top finish. The concrete was machine mixed 7 sand and gravel to 1 cement and wheeled in barrows about 175 ft. on the average.

Account 8308—Boiler Building—Steel Structure.

(See account 7308.2.)

There are in this building 35.03 tons of corrugated iron and 257 tons of structural steel.

Account 8308.5—Platforms and Brackets.

These were structural steel walkways installed after the boilers had been piped and bricked, furnished by the Kansas City Structural Steel Co. and erected by the Arizona Copper Co. construction force. There were 29.5 tons of material. The installation necessitated boring for connections and much removing of steam piping.

Account 8309—Waste Heat Boilers, Including all steel.

This account covers the cost of the 7 waste heat boilers with the steel framework of the settings erected. These boilers are class M No. 26 Stirling waste heat boilers, for 180 lb. pressure. They have 7,460 sq. ft. of total heating surface and occupy each a space of 16 ft. by 20 ft. 4 in. by 26 ft. 4¾ in. The waste heat from the flue, common to all the reverberatories, enters the front of the boilers at the top. (See Fig. 87.)

Account 8309.01—Waste Heat Boilers—Brickwork.

This account covers all red and fire brick and tile with mortar and lumber for scaffolds, as well as mason and carpenter labor entering into the bricking of the waste heat boilers. The unloading of the brick and handling from the pile to the boiler site are taken care of elsewhere. (See Fig. 87.)

Account 8309.02—Waste Heat Boilers—Unloading Brick.

This covers the cost of preparing the site, unloading and checking all brick used under this account.

Account 8309.03—Waste Heat Boilers—Painting.

When the boiler settings were warm, they were given one coat of mineral red in oil. This account covers the labor and material incident to this operation.

Account 8309.05—Waste Heat Boilers—Rehandling Brick.

This covers the cost of handling brick a distance of 250 ft. from piles to site of waste heat boilers in wheelbarrows.

Account 8309.10—Oil-fired Boilers, Including all Steel.

Same as account 8309. (See Fig. 88.)

Boilers were 3, Class M, No. 14 Stirling for oil firing, having each 4,017 sq. ft. of heating surface. Each boiler occupied a space of 10 ft. by 20 ft. 4 in. by 26 ft. 4¾ in. The oil burners are not here included, but 3 stacks are of 48-in. diameter each and 60 ft. high above damper frame, made of No. 10 and No. 8 steel. (See Fig. 88.)

Account 8309.11—Oil-fired Boilers—Brickwork.

Same as account 8309.01. (See Fig. 88.)

Account 8309.12—Oil-fired Boilers—Unloading Brick.

Same as 8309.02.

Account 8309.13—Oil-fired Boilers—Painting.

Same as 8309.03.

Account 8309.15—Oil-fired Boilers—Rehandling Brick.

Same as 8309.05, save distance was about 300 ft.

Account 8310—Superheaters—Waste Heat Boilers.

This account covers the cost of the material and the labor of installing 7 Foster superheaters for class M, No. 26 waste heat Stirling boilers.

7 Foster superheaters	$8,256.40
Miscellaneous	32.31
	$8,288.71

(See Fig. 88.)

Account 8310.10—Superheaters—Oil-fired Boilers.

This account covers the cost of the material and the labor of installing 3 Foster superheaters for class M, No. 14 oil-fired boilers.

3 Foster superheaters	$2,675.03
Miscellaneous	81.38
	$2,756.41

Account 8312—Miscellaneous Piping. Boilers and Reverb. Building.

This account covers the cost of material and installation of miscellaneous piping in the boiler and reverberatory buildings. The sizes are various. It is not valuable for unit costs.

Account 8312.1—Excavation.

This account covers excavation and backfill for a long deep trench. The material met with was red clay filled with boulders and sand and gravel. It was done with picks and shovels, and handled 300 ft. with wheelbarrows and slips. Two hundred feet of the trench were cribbed and lagged 20 ft. high. Much of the dirt had to be handled three times in removing it from the trench.

Account 8312.11—Feed Piping from Heating Plant to Feed Pumps.

This account covers the cost of the pipe, pipe conduit, insulating material and the labor incident to installing them in a trench running from the hot water heating plant to the boiler feed pump house back of the boilers. The conduit was ordinary vitrified 15-in. sewer pipe split in halves. The first half was laid in the trench, the joints cemented, followed by the laying of the 8-in. standard wrought iron pipe. About this the asbestos filler was packed and after each section of the conduit top was laid, the filler was stuffed in over the top of the 8-in. pipe to thoroughly cover it. The material account is segregated as follows:

557 ft. 15-in. J. M. sectional conduit	$2,273.47
557 ft. 8-in. wrought-iron pipe	374.49
Asbestos filler and miscellaneous	109.83
	$2,757.79

Account 8312.20—Feed Piping from Pumps to Boilers.

This cost is not valuable for unit purposes. It represents pipe fittings, pipe covering, paint, and the labor of erecting pipe and fittings, covering some of the pipe with insulation, and painting all pipe. The piping was about one steam and two electrical feed pumps at the boilers. It also covers a hot water line the length of the boiler building, a cold water line the same length with connections from each line to each boiler. The two main lines are 6 in. The connections to the boilers are 3 in. The hot water lines are covered throughout. The pipes are of standard strength and the fittings are extra heavy. The labor costs include also the manufacture of all pipe hangers. A further segregation of the material is as follows:

Pipe	$416.39
Fittings	2,408.89
Pipe covering	137.26
Hangers and miscellaneous	78.46
	$3,041.00

Account 8312.5—Blow-off Piping and Drum.

This cost is not valuable for unit purposes. It represents the cost of the material below and the labor of installing it. The blow-off piping runs about 10 ft. beyond the end of the boiler building and discharges there into a steel drum 4 ft. diameter by 4 ft. high, with an 8-in. diameter pipe riser. From the base of the drum it discharges into a sewer pipe nearby. The drum foundation is included in the concrete cost. A segregated material cost is as follows:

225 ft. 2½-in. standard wrought-iron pipe 25 ft. 8-in. standard wrought-iron pipe 20 ft. 2½-in. extra heavy non-rising stem gate valves 20 ft. 2½-in. asbestos packed cocks	$447.93
1 blow-off drum, 4 ft. diam. by 4 ft. high	79.84
	$527.77

Account 8313—Wiring Electrical Feed Pumps.

This covers the wiring of the two 40-h.p. motors of the feed pumps to the mains. The material was as follows:

2 circuit breakers	$31.70
Conduit and covering	85.20
Wiring and miscellaneous	60.99
	$177.89

Account 8313.1—Lighting for Reverb. and Boiler Building.

This represents the material and labor of hanging 104 drop lights in the boiler and reverberatory building.

8 tungsten lamps, 40 watt 96 carbon lamps, 16 c.p. 145 ft. brewery cord 2,720 ft. 1-in. conduit Wire, switches, etc.	$473.19

Account 8314—Slag Launders.

All the material cost and labor installation of these reverberatory slag launders and two converter slag launders are here included. The reverberatory slag launders at the slag end of the furnace consist of a settler and spout. The settler is about 6 ft. by 3 ft. by 2 ft., made of ⅜-in. steel plate and 3-in. by 3-in. angles. The spout leading from the settler to the slag cars is of cast iron 6 ft. 6 in. long and 1 in. thick. The converter slag launders are built of cast iron about 1½-in. thick on the average, in 4-ft. 6-in. sections and are 24 ft. 6 in. long. They are set up aloft in the converter building on a structural steel frame made of 10-in. 15-lb. I's and 2½ by 2½ by $\frac{5}{16}$ angles attached to the framework of the building, and lead into the top of the reverberatory furnaces.

Account 8315—Matte Launders.

These launders lead from small settler boxes at the tap holes of the reverberatories along the dirt floors in which they are set to a height of 10 to 12 ft. above the converter building floor, where they discharge into properly located matte pots. The small boxes are of ⅜-in. plate with 3 by 3-in. and 2½ by 2½-in. angles. The launders are of cast iron average 2⅜ in. in thickness. Here is included the cost of the above materials together with the labor of installing them.

Account 8316—Six No. 14 Wilgus Oil Systems.

This account covers the cost of 6 Wilgus oil pumps, asbestos covering for portions of these pumps, the labor of installing the pumps, the labor of thoroughly overhauling them, required because of the unsatisfactory condition existing in the leaking steam heating coils, and the labor of applying the asbestos covering. The 5½-in. by 3½-in. by 5-in. duplex oil pumps were set directly on the concrete floor in front of the oil-fired boilers.

Account 8317—Two Electrical Feed Pumps.

These pumps located back of the boilers were lowered into the 13-ft. pit onto their foundations and set ready for piping connections. They are two vertical triplex, 8-in. by 10-in. Aldrich, electrical driven pumps each attached with flexible couplings to a 40-h.p. motor. The cost covers the material segregated below and the labor of installing the same:

	Factory	Freight	Clifton
Two 40-h.p. motors	$1,700.00	$24.44	$1,724.44
Two vertical triplex pumps	2,794.00	547.07	3,859.07
Spare parts for pumps	518.00		
Miscellaneous			50.46
			$5,633.97

Account 8317.1—One Steam Feed Pump.

Here is given the labor of installing and the material cost of one 10-in. by 6-in. by 12-in. duplex boiler steam feed pump. This pump is located next to the two electrically driven Aldrich pumps of 8317.

Account 8317.2—Crawls and Chain Blocks in Feed Pump House.

This gives the cost of delivering and hanging in place in the feed water pump house two 2-ton steel plate crawls for lower flange of 8-in. I-beam, one 2-ton duplex chain block for 16-ft. lift, together with the material cost as segregated below.

Two 2-ton crawls	$53.74
One 2-ton chain block	58.31
	$112.05

Account 8318—Fettling System.

Here is given the cost of installing fettling tracks and cars over the three reverberatories. A structural steel frame to support an 18-in. gauge car and walkway was erected along each side of each reverberatory, being attached to the steel frame of the building. The cost of this material as given below and the labor of installing the same are here covered:

Three 18-in. gauge bottom dump cars with Hyatt roller bearings	$196.83
3 steel turn plates ¼ in. by 72 in. by 138 in.	93.24
18.25 tons fabricated structural steel and rail	1,492.66
2 by 12 lumber for walkways	18.48
Power for riveting and miscellaneous	141.87
	$1,943.08

CONVERTER PLANT

Account 8401—Excavation.

This was a large slice, similar to side hill work, through red clay and boulders into sand and gravel tightened with caliche. It was shaken up with powder, plowed, fresnoed through a trap into narrow-gauge side dump cars and conveyed 1,000 to 2,000 ft. by a narrow-gauge locomotive.

Account 8402—Foundation.

This was a big wall same as under 8301, with about 30 piers 5 ft. by 6 ft. by 8 ft. deep. The concrete was machine mixed, 7 parts sand and gravel to 1 cement, transported 75 ft. in concrete buggies and wheelbarrows. About 50 per cent. of the vertical surfaces was formed. Each pier had four 2-in. anchor bolts 6 ft. long.

Account 8403—Converter Building—Steel Structure.

(See account 7308.2). There is in this building 94.01 tons of corrugated iron and 689.85 tons of structural steel.

Account 8404—Converter Stands—Excavation.

This excavation was small rectangular cuts in sand and gravel, made with picks and shovels and handled in wagons 900 ft.

Account 8404.1—Converter Stands—Foundation.

This concrete was the same mix as 8402, but 100 per cent. of its vertical surfaces was formed. It was hauled 300 ft. to place in dump wagon and cars.

Account 8405—Converter Stands and Shells.

This account covers the cost of the material noted below, together with the labor required to install the stands, put together the parts of the converters, erect motors, controllers, solenoid brakes and attach blast connections. The unloading of this material from the cars is in account 8411. The shells were 12 ft. in diameter, of the Great Falls type, having each 28 tuyères of 1½ in. diameter, extra heavy pipe.

Three cast-iron converter stands, with 50-h.p. motors, brakes, controller, etc.	$9,801.01
Four 12-ft. converter shells	12,115.46
Blast connections, valves, etc.	321.81
	$22,238.28

(See Figs. 126 and 127.)

Account 8405.01—Repairs to No. 2 Stand.

When No. 2 converter shell was let into place by the crane, it was allowed to fall a short distance and cracked the cast-iron stand. The stand was then taken off its foundation and patched with a steel plate.

Account 8405.10—Converter Shells—Brick Lining.

This account covers both the labor and material incident to lining four converter shells and tops with magnesite brick. In bottom of each shell there is an average of 9 in. of silicate of soda and burnt magnesite beneath the brick. Around the sides there are 4½ in. of this material laid in behind the brick. The top is laid with brick only. The material required for one converter is as follows:

28 sacks (286 lb. each) of magnesite cement.
50 sacks (234 lb. each) of burnt magnesite.
5 bbl. (635 lb. each) of silicate of soda.
4,385 magnesite brick of various shapes.

Account 8405.11—Converter Shells—Unloading Brick.

This covers the cost of the shed together with the checking, unloading and piling of all magnesite brick, cement, magnesia and sodium silicate.

Account 8406—Cranes.

This covers the cost of two 40-ton Morgan cranes and the labor of installing them on the craneway, and putting together the equipment ready for operation. It does not include the wiring. They were hoisted in place on the craneway by the use of two erecting engines. These cranes are of 40-ton capacity, have four motors, span 55 ft. from rail to rail, and are rigged for a 50-ft. lift. Each crane has a 15-ton auxiliary hoist. (See Fig. 90.)

Account 8406.1—Wiring Cranes.

This cost is not valuable as it represents 30 per cent. more labor than should have been spent. The cranes were wired twice because the first time was done improperly. The wiring is largely in conduits. Here too is the cost for the trolley lines from which the cranes take their power.

Account 8407—Clinkering Machines.

These two machines are set 24 ft. above the floor of the converter building on structural steel supports. The steel supports are a part of the converter building and have been costed in that account. The main body of the machine, the mixer, is the frustum of a cone 13 ft. 6 in. long, whose head end is 5 ft. diameter and whose discharge end is 9 ft. 6 in. diameter. It is made of ¾-in. steel plate, lined with 1-in. cast-iron liners. The whole is mounted on trunnions operated by a 50-h.p. motor. The ladle which feeds the converter slag into the head

end is 60 cu. ft. capacity and is tilted by a screw operated by a 15-h.p. motor.

The feeder which lets siliceous ore into the head end to agglomerate with the slag extends from the silica bins to a pipe discharging into the dropping stream of slag. It is a screw conveyor 4 ft. 9½ in. long. Each machine has a hood connected to a steel flue 2 ft. 6 in. diameter by 36 ft. 8 in. long, leading into the converter dust chamber.

The machinery for two machines enumerated above cost	$11,872.82
Two 50-h.p. motors as above	828.61
Two 15-h.p. motors as above	820.16
2 brakes for ladle tipping motor	176.51
2 traveling switches for brakes	136.44
2 circuit breakers	102.80
Miscellaneous	44.60
	$13,981.94

This cost includes the price of the machines and the cost of installing them.

Account 8407.01—Clinkering Machines—Alteration No. 1.

The teeth on the drive gears had to be chipped off and trued up so as to mesh properly.

Account 8407.02—Clinkering Machines—Alteration No. 2.

Account 8407.03—Clinkering Machines—Electrical Alterations.

Account 8407.1—Wiring Clinkering Machines.

This covers the labor and material of wiring the two 50-h.p. motors and two 15-h.p. motors operating the clinkering machines.

Account 8409—Wiring for Converter Control.

Account 8409.1—Lighting.

This covers the labor of installing the material incident to lighting the converter building, as well as the material itself. There were used thirty-four 16-c.p. carbon lamps, twenty-six 250-watt tungstens, 440 ft. of No. 8 and No. 12 weatherproof wire exposed, 880 ft. of No. 12 wire in ¾-in. conduit, 684 ft. of No. 12 wire in 1-in. conduit.

Account 8410—Air Pipe from Power House—Excavation.

This covers the cost of digging a trench through sand, gravel and big boulders for a 24-in. pipe, with pick and shovel and backfilling the same.

Account 8410.1—Air Pipe from Power House—Laying.

This covers the cost of the material segregated below and the labor of installing it. The pipe was placed underground and ran from the power house to connect with all of the converters. It was built to carry air

under 12 lb. pressure, of No. 8 U. S. gauge plate, riveted, tested for 25 lb. pressure and painted with asphaltum paint. It was made in 30-ft. sections and fastened together with forged steel flanges.

400 ft. 24-in. pipe, 10 in. cast-iron nozzles, tees and ells	$1,332.70
22 ft. 10-in. pipe and two 10-in. flanges	27.54
Two 24-in. cast-iron gate valves	415.25
Three 10-in. cast-iron gate valves	138.55
Miscellaneous	127.85
	$2,041.89

(See Fig. 89.)

Account 8411—Ladles, Boats, Bails, Tools, Etc.

This account covers the cost of the material segregated below, the labor of unloading it, the labor of unloading material in account 8405, and the cost of material and manufacture of several converter collar pullers, as well as alterations upon the slag ladles and scrap boats.

	Factory	Freight	Total
One 7-ft. by 7-ft. slag boat complete with chain	$362.00	$38.00	$400.00
2 converter scrap boats 9 ft. by 2 ft. 3½ in. high by 2 ft. 5 in. wide	512.60	28.02	540.62
3 cast-steel slag ladles	745.90	283.04	1,028.94
2 cast-steel matte ladles (20 tons capacity)	1,152.45		
1 bail	400.00		
1 bail	200.00	519.11	2,371.56
1 pattern	100.00		
4 chains and converter lifting devices			340.63
1 cast-iron skull breaker			124.99
Miscellaneous material for collar, etc			125.65
			$4,932.39

Account 8413—Casting Machines—Excavation.

This covers 2 deep rectangular cuts in sand, gravel and big boulders with pick and shovels. It was loaded into carts and hauled 600 ft.

Account 8414—Casting Machine—Foundation.

The foundation for each machine consisted of a rectangular sump with plain concrete floor enclosed by reinforced concrete retaining walls. The walls were about 6 in. thick, 8 ft. high, reinforced with ⅝-in. and ¾-in. rods. The concrete was machine mixed, 5 parts sand and gravel to 1 cement, hauled in cars 150 ft., dumped, and handled to site in wheelbarrows 150 ft. 100 per cent. of the vertical concrete surfaces was formed.

Account 8415—Casting Machine—Cost and Erection.

This account covers the cost of all the material composing 2 casting

machines, and all the labor required to erect on their foundations ready to operate. Each machine has a steel cradle to receive a ladle of molten copper. This cradle is controlled from a pulpit and is tipped by the power from a 20-h.p. motor. It is set high enough to pour into a casting spoon of 1½-in. cast iron whose approximate dimensions are 2 ft. wide by 3 ft. 6½-in. long, and from 7 in. to 1 ft. 5½-in. deep. This casting spoon pours into the molds which are attached to a heavy steel conveyor. The molds are 39 in number, made of 2½-in. cast iron reinforced with $\frac{5}{16}$-in. perforated plate. Their inside dimensions are 2 ft. 4 in. by 1 ft. 6¼ in. by 3¼ in. deep. From the pulpit, by use of power from a 20-h.p. motor, the conveyor with the molds moves along under a spray of water from needle holes in pipes placed above them until they reach the end of the conveyor, where a device in the bottom of the molds loosens the ingots, allowing them to drop into a tank of water. This bosh is made of $\frac{5}{16}$-in. plate, 3 by 3 and 4 by 3 angles. It is 7 ft. wide, 23 ft. 5¾-in. long, and varies in depth from 7 ft. 10 in. to 2 ft. 10 in. The copper bars are removed from here by a steel drag conveyor operated by an 11-h.p. motor, controlled from the pulpit. When the bars leave the bosh and fall onto the striking plate they are handled by a radial crane whose moving end travels on a 40-ft. curved I-beam. Along the radial crane beam travels a small air hoist capable of picking up 1 ton. It operates under an air pressure of 16 lb. A jib crane is so located, attached to a building column, that it can handle the molds for removing and replacing. It has a 3,000 lb. capacity triplex block and 8-in. I-beam trolley. Below is a segregated material list:

2 casting machines	$18,657.89
Two 11-h.p. and four 20-h.p. motors	2,933.88
2 jib cranes	327.22
2 radial cranes	1,167.91
2 traveling switches	135.75
2 brakes for ladle tipping motors	176.51
4 circuit breakers	103.50
Molds, etc	708.55
	$24,211.21

(See Figs. 128 and 129.)

Account 8415.1—Casting Machine—Repairs.

Account 8416—Loading Platform—Excavation.

Same as 8413, except that it was not hauled away.

Account 8416.1—Loading Platform—Foundation.

This was a low retaining wall of gravity section 300 ft. long, machine mixed, 7 sand and gravel to 1 cement, transported in cars 150 ft., by wagon 350 ft. and by wheelbarrow 70 ft. One hundred per cent. of its vertical surface was formed.

Account 8416.11—Loading Platform—Floor.

This was a plain concrete floor mixed and handled as above, with a ½-in. finish of 2 sand to 1 cement. There were no joints in the concrete. The finish was troweled smooth.

Account 8416.2—Loading Platform—Backfill.

Behind the 300-ft. wall 8416.1—sand and gravel was backfilled. The material lay 8 to 10 ft. from the wall.

Account 8416.3—Loading Platform—Striking Plates.

Two striking plates, one at each casting machine are placed so that the copper ingots discharged from the casting machine elevator fall directly upon them. They were made by setting 4 by 4 by 4½-in. wood blocks dipped in hot tar and placed on end upon a concrete base. Over the blocks a steel plate 6 ft. ½ in. by 9 ft. 10 in. by ½ in. was laid and secured by 16 ¾ by 2½-in. bolts, grasped by cast-iron fasteners set in concrete below.

283 ft. b.m. lumber	$8.59
2 steel plates	99.19
32 cast-iron fasteners and bolts	18.91
	$126.69

Account 8417—Hoods and Smoke Boxes.

This account covers all the material of the converter hoods, smoke boxes, flues leading to converter dust chamber, together with the labor of erecting them. It likewise includes removing the stacks 4 ft. in diameter, making new ones 5 ft. in diameter and erecting them, together with change required to put large doors in the back of the boxes. The smoke boxes, of which there are three, are made of ⅜-in. plate, and 4 by 4 by ¾ angles. They are 16 ft. high and about 9 ft. in diameter. The hoods, of which there are three, are made from ½-in. plate and 4 by 4 by ⅜ angles. They hang on the front of the smoke boxes and direct the gases into the flues. The original stacks, of which there were three connecting the smoke boxes and the converter dust chamber, were 4 ft. diameter and 26 ft. long, made of $\frac{3}{16}$-in. plate. They were replaced by similar ones 5 ft. in diameter. (See Fig. 92.)

Account 8417.1—Hood to Protect Converter Operator.

Only one of these was made. Three-sixteenth inch plate was used. The dimensions are 7 ft. 2 in. by 7 ft. 2 in. by 7 ft. 8 in. high, one end is open. The account covers material used, fabrication and erection.

Account 8418—Spouts, Gates and Hoppers at Silica Ore Bins.

This account covers the material cost of the gates with operating devices, the 10-in. pipe chutes and the labor of erecting same, together

with the labor of erecting the hoppers. The hoppers were furnished by the Kansas City Structural Steel Co., and are costed with the building. The hoppers are situated below the silica bins, above the converters, and by a spring device and pointer indicate to an operator on the ground when they have been filled to the desired amount. The gates allow the material to flow through a 10-in. pipe chute directly into each converter. These chutes can be turned aside from the converter mouth by a chain, wheel and gear so as not to interfere when out of use. (See Fig. 93.)

Account 8419.1—10-ton Bullion Scales—Excavation.

The excavation consisted of small cuts made in sand and gravel with pick and shovel and cast to one side.

Account 8419.2—10-ton Bullion Scales—Foundations.

This concrete was cast plain in low 8-in. thick walls about a pit 4 ft. by 6 ft. in plan. The mix was machine made, 6 sand and gravel to 1 cement, and transported a distance of 1,900 ft. in wagons. Seventy-five per cent. of the walls' vertical surface was formed.

Account 8419.3—10-ton Bullion Scales—Cost and Erection.

This represents the cost of the scales and the labor of installing them. The scales were pit pattern, 10-ton copper bullion class, with type registering beam weighing to 1 lb. They came complete with all necessary structural steel framework and cast-iron platform plate.

Account 8419.4—10-ton Bullion Scales—Scale House.

This is a shed roof building without sides about 16 ft. by 20 ft. The roof is of 1-in. sheathing, covered with composition roofing. It was painted 2 coats of oil and lead.

Account 8425—Conveyor No. 15.

(See account 7405.)

Conveyor No. 15 is a 20-in. belt, making a conveyor 165 ft. long, running perfectly flat, operating at a speed of 300 ft. per minute, capable of handling 100 tons per hour. It receives material from conveyor 14 and delivers it to the silica bins of the conveyor building through an automatic tripper. The account is segregated as follows:

Belt	$664.19
Robins material	1,231.91
Centrifugal switch	36.20
Lumber (decking, etc.)	67.05
Spout from No. 14 to No. 15	31.21
29 ft. 6 in. of 5-in. d.l. drive belt	20.82
7½-h.p. motor	138.75
Miscellaneous	61.34
	$2,251.47

(See Fig. 91.)

Account 8426.1—Wet Pan—Excavation.

Account 8426.2—Wet Pan—Foundation.

This concrete was hand mixed. Owing to some conditions not satisfactorily ascertained the concrete did not set. This necessitated its being put in twice. The yardage is that of one installation and the cost two. The mix is 7 to 1.

Account 8426.3—Wet Pan—Cost and Erection.

This mill was installed to furnish "mud" for the converters and reverberatories. The account covers the material segregated below and the labor of installing the same.

One 5-ft. wet pan; size of mullers 36 in. by 4½ in.; pulley 34 in. by 10 in.; 4-arm type friction clutch	$634.34
One 15-h.p. motor, 860 r.p.m. squirrel-cage with starting compensator	248.45
One 18 by 10 solid hub cast-iron pulley	15.47
One 38 by 7 solid hub cast-iron pulley	24.71
Two 2 1/16 by 24-in. drop hangers	23.48
11 ft. 2 in. 2 1/16 shafting, collars, etc	13.10
30 ft. 6-in. double leather belting	25.39
35 ft. 9-in. double leather belting	44.42
Miscellaneous	20.74
	$1,050.10

Account 8426.4—Wet Pan—Bins and Spout.

This bin with spout was made in the smelter shops, using ¼-in. steel plate. It has a capacityof 260 cu. ft. The account covers the material used, labor of fabrication and erection.

CONVERTER DUST CHAMBER

Account 8421—Excavation.

This account covers the making with pick and shovel of small cut for a retaining wall, and digging a number of small pier holes. The material was red clay and stones, running into sand and gravel, which was loaded into carts and hauled 600 ft.

Account 8422—Foundation.

This concrete was cast as piers about 4 ft. by 4 ft. by 5 ft. about 45 per cent. of whose vertical surface was formed. It was mixed in a machine, in the proportions of 7 sand and gravel to 1 cement, transported by cars and wheelbarrows 200 ft. The pier tops were finished to a perfect elevation to receive structural steel columns.

Account 8423—Steel Structure.

(See account 7308.2.) This structure contained 228.18 tons of structural steel and 10.12 tons of Keystone plate roofing. (See Fig. 94.)

Account 8423.01—Wire Baffles.

(See account 8123.01, for description.) This account covers the cost of the material below and the labor incident to its erection.

14,365 lb. No. 10 steel wire baffles	$670.34
8,500 lb. ⅜-in. steel chains, 2-in. links	430.91
Miscellaneous	0.70
	$1,101.95

(See Figs. 94 and 95)

Account 8423.1—Tile Work.

This is identical with 8123.1.

Account 8423.11—Unloading Tile.

This is identical with 8123.11.

Account 8424—Iron Doors and Frames.

This covers the cost of the cast-iron doors, etc., set in the tile work of the converter dust chamber. The labor represents hauling the same to the site. The labor of setting is included with the tile work.

5 cast-iron peep doors and frames, doors 4 ft. 6 in. by 2 ft. 6 in..... $158.93

Account 8428—Smoke Box Track.

This is a track back of the smoke boxes for the converters. The material is second hand, picked up from construction equipment. The account is of no value.

CONVEYING SYSTEM

Account 8501—Excavation.

This covers excavation made at various times for piers and trenches for walls to support the conveying system structures. The ground was mostly red clay and boulders, sometimes sand and gravel. The excavating was done with pick and shovel and the material cast to the side of the cuts.

Account 8502—Foundation.

This account covers plain concrete cast in a great many piers, and reinforced concrete cast in a shape to make two long tunnels through which conveyors 11 and 14 rise from below conveyors 10^1 and 10^2 located under the bunker bins. The tunnels are 6 ft. by 6 ft. with 12-in. walls, reinforced with ½-in. and ¾-in. rods, spaced 6 in. About 80 per cent. of the vertical surfaces was formed. All concrete was machine mixed in different proportions and transported variously to the many different situations.

Account 8503—Steel Structure.

(See account 7308.2.) There were here used 30.94 tons of corrugated iron, and 180.79 tons of structural steel. These structures are elevated steel conveyor ways.

Account 8504—Woodwork.

This account represents the labor and material of flooring the steel conveyor ways for conveyors 3, 4, 5, 6, 11 and 14. The lumber used was 2 by 12 S1S2E No. 1 merchantable Oregon pine. On No. 14 the 2 by 12's were rabbeted. Considerable cutting was done to frame about conveyor steel frame supports. This cost includes also attaching nailing strips to the steel work to which the flooring was nailed. (See Fig. 96.)

Account 8504.1—Floor Battens.

This account covers labor and material incident to nailing battens beneath the floor boards of conveyors 3, 4, 5, 6 and 11. The lumber here used was not rabbeted.

Account 8505—Conveyors No. 3, 4, 5, 6, 11 and 14.

(See account 7405.) Conveyor No. 3 has a 20-in. belt, making a conveyor 182 ft. 5½ in. long, rising 46 ft., operating at a speed of 250 ft. per minute, with a capacity of 150 tons per hour. It conveys concentrates from No. 2 belt into the sampling mill.

Conveyor No. 4 has a 20-in. belt, making a conveyor 220 ft. 9 in. long, rising 64 ft., operating at a speed of 250 ft. per minute, having a capacity of 100 tons per hour. It takes crushed ore from the crushing plant to the top of the sampling mill.

Conveyor No. 5 has a 20-in. belt, making a conveyor 127 ft. long, having a rise of 26 ft. 4 in., operating at a speed of 250 ft. per minute, with a capacity of 150 tons per hour. It carries the fines from the sample mill on their way to the beds.

Conveyor No. 6 has a 20-in. belt, making a conveyor 113 ft. 8 in. long, having a rise of 25 ft. 6 in., operating at a speed of 250 ft. per minute with a capacity of 100 tons of ore per hour. It carries material from the sample mill on its way to the bunker bins.

Conveyor No. 11 has a 20-in. belt, making a conveyor 369 ft. 8½ in. long, having an 87-ft. rise, operating at a speed of 300 ft. per minute with a capacity of 100 tons per hour. It conveys the product from conveyor 10^1 to conveyor 12 at the roasters.

Conveyor No. 14 has a 20-in. belt, making a conveyor 271 ft. 5 in. long, having a 71-ft. rise, operating at a speed of 300 ft. per minute with a capacity of 100 tons per hour. It takes ore from conveyor 10^2 beneath the bins to conveyor 15.

The material for these conveyors somewhat segregated is as follows:

	Factory	Freight	Clifton
Belt	$5,351.03	$365.88	$5,716.91
Robins material	4,290.86	530.49	4,821.35
Extra pulley	.....	.....	38.00
4 centrifugal switches	136.00	8.81	144.81
One 10, one 15, and four 20-h.p. motors	.....	.....	1,309.68
5 overload releases	65.00	6.25	71.25
Miscellaneous, decking, etc	.....	.....	403.62
			$12,505.62

Account 8505.1—Chutes.

This cost is of no value.

Account 8505.2—Guides.

The belts in the conveying system could not be made to run true on the troughing idlers. To overcome their riding out of position long boards were fixed at the sides of the belts to guide and keep them in position. These boards were picked up about the plant and the cost represents only the labor of installing them.

Account 8505.3—Weightometer.

This account covers the cost and labor of installing a Merrick weightometer on conveyor 11. The weightometer is installed on a 20-in. inclined conveyor belt with a speed of 300 ft. per minute, whose angle of inclination is 13 degrees 28 minutes and whose troughing idlers are 4 ft. on center lines. The belt has a normal capacity of 100 tons per hour.

Account 8506—Lighting.

This represents installing the following lights:

33 drops
1,285 ft., No. 12 weatherproof wire
120 ft. of conduit

CHIMNEY

Account 8601—Excavation.

This was a deep hexagonal cut made through clay, caliche and well into sand and gravel containing big boulders. The material was loosened with picks, slipped out with fresnos, dumped through a trap into carts and hauled 2,700 ft.

Account 8602—Foundation.

This was a very large block of concrete cast in a hexagonal shape 20 ft. deep and 50 ft. inside least diameter. In the bottom of the block 3 layers of 1-in. rods laid 1 ft. on centers were placed. The mixture was machine made, 8 parts sand and gravel to 1 cement, using lots

of large rock. About 40 per cent. of the vertical surface was formed. The concrete was transported in cars 100 ft.

Account 8603—Brickwork.

The stack was contracted erected by the Alphons Custodis Chimney Construction Co. It is 300 ft. high, 26 ft. 8 in. inside diameter at the base and 22 ft. at the top. The average thickness of the walls is about 24½ in. Every 25 ft. inside the stack is corbelled out to hold the lining of radial perforated fire brick, laid in acid-proof mortar. The base of the stack is of red brick and the round portion is of perforated radial blocks. The outside upper 75 ft. of the stack were pointed with acid-proof mortar. There was used in the construction:

138,000 lb. lime	652 tons wire cut brick
290 bbl. cement	56 tons wedge brick
1,638 tons radial brick	100 bbl. acid-proof mortar

The cost here given includes constant inspection by the Arizona Copper Co. organization. (See Fig. 97.)

REVERBERATORY FLUE

Account 8611—Excavation.

This covers the excavating of some long deep trenches for footings and a large amount of back filling. It was done in red clay and gravel with picks and shovels. The back filling was wheeled 25 ft. to place and tamped in 5-in. layers.

Account 8612—Foundation.

This concrete was cast in 2 long reinforced concrete cantilever type retaining walls. The walls were 12 in. at the top, 14 in. at bottom, and 5 ft. high. One-half-inch and ¾-in. rods, spaced 6 in. centers, were used. The mixture was machine mixed in the proportion of 5 sand and gravel to 1 cement, transported in wagons, wheelbarrows and concrete carts 250 ft. to place. Ninety-five per cent. of the vertical surface of the concrete was formed.

Account 8613—Brickwork.

This account is similar to others of the same nature, including cost of tile, mortar, scaffolds, and the labor of masons, their helpers, and carpenters. (See Figs. 98 and 100.)

Account 8613.01—Unloading Brick.

This covers the cost of preparing site, unloading tile, and the checking of same.

Account 8614—Steel Structure.

(See account 7308.2.) There were 32 tons of structural steel used here and 9.61 tons of Keystone plate roofing. (See Fig. 99.)

Account 8614.1—Clean Out Doors.

This covers the cost of labor of altering and material in the clean out doors and frames for this flue

18 cast-iron frames and steel plate doors, $\frac{3}{16}$ in. by 16½ in. by 2 ft. 2½ in	$128.27
¼-in. sheet steel and miscellaneous	25.34
	$153.61

Account 8614.2—Calking Roof.

This account covers the labor and material of making as nearly air-tight as was possible the roof to this flue. Asbestos wicking was calked into all the bad joints.

CONVERTER FLUE

Account 8621—Excavation.

This was a small amount of excavation for a number of piers through red clay with boulders and sand and gravel. It was done with pick and shovel, the dirt being cast to the sides of the holes.

Account 8622—Foundation.

These foundations were 14 plain concrete piers about 4 ft. 6 in. by 4 ft. 6 in. by 5 ft. The concrete was machine mixed, about 7 parts sand and gravel to 1 cement, and transported 200 ft. in wheelbarrows and concrete carts to place. Seventy-five per cent. of the vertical surfaces was formed.

Account 8624—Steel Structure.

(See account 7308.2.) 81.99 tons of structural steel were used here. (See Fig. 101.)

ROASTER DUST CHAMBER FLUE

Account 8626—Excavation.

This excavation covers the cuts for a number of piers through red clay containing boulders, made with pick and shovel and thrown to one side of the excavation.

Account 8627—Foundation.

This concrete was cast in 12 piers about 4 ft. 6 in. by 4 ft. 6 in. by 5 ft. It was plain concrete, machine mixed in proportions of 7 sand and gravel to 1 cement, and was transported to place 200 ft. in cement cars and wheelbarrows. Seventy-five per cent. of the vertical surface was formed.

Account 8628—Brickwork.

This is the same as 8123.10. (See Figs. 99, 100, 102, and 104.)

Account 8628.01—Unloading Tile.

This is the same as 8123.11.

Account 8629—Steel Structure.

There were 85.21 tons of structural steel used here and 9.25 tons of Keystone plate roofing. The flue is 6 ft. 6 in. by 12 ft. in cross section and connects the roaster dust chamber with the stacks. (See Fig. 102.)

BOILER AND BLACKSMITH SHOP

Account 8701—Excavation.

This excavation involved making a 6-ft. slice to get the proper grade for the building site, together with piers and small wall excavation. It was plowed and slipped away in fresnos 400 ft.

Account 8702—Foundations.

These foundations were the small walls and piers for the brick and steel column supports. The concrete was plain, hand mixed in the proportions of 6 sand and gravel to 1 cement, and handled 100 ft. in wheelbarrows to the forms. Fifty per cent. of the vertical surface was formed. This was the first concrete cast at the smelter.

Account 8703—Steel Structure.

There were 32.72 tons of structural steel used in the framework of the building. (See Fig. 105.)

Account 8703.1—Doors, Windows and Frames.

This account covers the purchase price of all doors, windows, their frames, lintels and glass. It also covers the labor of installing the steel lintels which run from building column to building column; the erection of the steel door and window frames; the erection of the steel sash and doors; and the glazing of these doors and windows. After the lintels had been framed in, the tile work brought up to sill base and the sill set, the frames were put in place, bolted to the lintels and tied by rods back to the building columns. When the frames had been entirely bricked in, the steel sash were bolted in place and later glazed. A segregated material list is as follows:

Thirteen 11 ft. 7 in. by 12 ft. ¼ in. steel sash 63 lights, 2 mullions, with 3-6 light ventilators, not glazed.

One 10 ft. 3 in. by 12 ft. ¼ in. steel sash 56 lights, 1 mullion, no ventilators, not glazed.

One 10 ft. 3 in. by 12 ft. ¼ in. steel sash, 48 lights, 1 mullion, no ventilators, not glazed.

One 4 ft. by 9 ft. steel sliding door, with six 14 in. by 20 in. lights, not glazed, lower panels steel plate.

One 8 ft. by 9 ft. steel sliding door, with eighteen 14 in. by 20 in. lights, not glazed, lower panels steel plate.

One 14 ft. by 20 ft. Kinnar steel rolling door.

One 10 ft. by 10 ft. Kinnar steel rolling door.
Eleven 14-ft. 10-in. lintels built up of 8-in. channels.
Two 13-ft. 4-in. lintels built up of 8-in. channels.
Two 11-ft. 6-in. lintels built up of 8-in. channels.
One 10-ft. 4-in. lintel built up of 8-in. channels.
One 11-ft. 6-in. lintel built up of 8-in. channels.
850 lights 14 in. by 20 in., ⅛ in. factory ribbed glass,
164 lights 13½ in. by 19½ in. factory ribbed glass.
82 lights 14 in. by 19½ in. factory ribbed glass.
44 lights 13½ in. by 20 in. factory ribbed glass.
Steel windows and door frames for above made of two 3½ by 2½ by ¼ angles.
(See Fig. 106.)

Account 8703.11—Concrete Sills.

This account covers the labor and material used to make the following list of concrete sills. The sills were made 3 parts sand and gravel to 1 cement, cast in collapsible molds and later finished. Three ⅝-in. rods are used in each sill.

11 sills, 8½ in. by 10 in., 14 ft. 10 in. long.
1 sill, 8½ in. by 10 in., 11 ft. 6 in. long.
1 sill, 8½ in. by 10 in., 8 ft. 6 in. long.
2 sills, 8½ in. by 10 in., 12 ft. 2 in. long.

(See Fig. 21.)

Account 8703.2—Tile Walls.

This cost includes the cost of tile, mortar and scaffolds, together with the mason and carpenter labor used to build the walls. The walls were non-bearing 8 in. thick, built of hollow tile, laid in between the steel building columns. The mortar used was 1 cement, 1 lime and 5 sand.

Account 8703.21—Unloading Tile.

This covers the cost of preparing site, unloading, and checking quantity of tile.

Account 8703.22—Coping.

This covers the cost of labor and material incident to coping the walls at the top, beneath the roof. A two by four was bolted to the top course of tile and another to the underside of the roof. These were lathed across with metal lath and plastered with cement mortar. (See Fig. 17.)

Account 8703.30—Roof.

This account covers the cost of the material and labor incident to roofing the boiler and blacksmith shop. Oregon pine sheathing, 2 by 8, surfaced, tongued and grooved, was nailed to strips bolted to the purlins. Over this 3-ply asbestos roofing paper was laid. (See Fig. 17.)

Account 8703.31—Ventilators.

This covers the cost of labor and material incident to installing three 48-in. Burt ventilators on the peak of the boiler and blacksmith shop roof.

The ventilators were skidded up onto the roof with hand tackle along a runway, bolted to the purlins and flashed. (See Fig. 18.)

Account 8703.4—Dirt Floor.

This account covers the labor incident to bringing the dirt floor of this building to the required grade. The dirt was wheeled in and tamped in 3-in. layers.

Account 8703.5—Benches.

This account covers the labor and material of making from time to time benches, racks and the like used in this shop.

Account 8703.6—Painting.

This covers the cost of painting all the steel sash one coat of "turkey red," and the woodwork, namely, the under side of the roof two coats of white lead and linseed oil, cream color.

Account 8704—Crane.

This covers the purchase of the crane listed below, the labor of overhauling and erecting it.

One 3-ton hand power traveling crane, chain block transfer type 18-ft. span, complete with roller bushed geared trolley and provided with 3-ton triplex chain block for 13 ft. lift	$378.35
Miscellaneous	60.06
	$438.41

Account 8705—Tools.

This account covers the purchase price of the tools enumerated below and the labor required to install them.

	Factory	Freight	Clifton
1 No. 2 punch and shear, Hilles & Jones	$1,530.00	$435.00	$1,965.00
1 No. 0 bending rolls	580.00	75.00	655.00
One 1,100-lb. stm. hammer, Niles-Bement-Pond Co.	1,015.00	408.00	1,423.00
One blower, size 5, type D, American Blower Co.; one 5-h.p. 440-volt, 3-phase, 60-cycle 1,720-r.p.m. motor	160.00	19.90	179.90
1 No. 5 swage block	.....	.....	35.08
1 Peter Wright anvil, weight 497 lb	.....	.....	70.57
10 in. galv. iron pipe and connections	.....	.....	106.63
3 sheets steel, ½ in. by 48 in. by 120 in	.....	.....	16.02
One 2-in. heading, upsetting and forging machine, Acme Machinery Co	2,790.00	440.70	3,230.70
1 Sisco anvil, 407 lb	.....	.....	46.60
1 Hay Budden anvil, 420 lb	.....	.....	48.10
40 ft. of 6-in. I-beam	.....	.....	12.62
Castings	.....	.....	41.00
Miscellaneous	.....	.....	29.14
			$7,859.36

Account 8706—Shafting, Pulleys, Belting.

This account covers the purchase price of the list of material below and the labor of installing the same, and the necessary wooden bridge trees.

33 ft. of $2\frac{7}{16}$-in. and 18 ft. of $2\frac{11}{16}$-in. shafting.
5 pulleys, varying from 26 in. to 52 in. with bearings and hangers.
1 length of 8-in. double leather belt, 104 ft. long.
1 length of 6-in. double leather belt, 140 ft. long.

(See Fig. 19.)

Account 8707—Motor.

This account covers the purchase price of the material below and the labor of installing it. This motor furnished the power for the boiler and blacksmith shops.

One 20-h.p. 440-volt, 3-phase, 60-cycle, 850-r.p.m. motor.

Account 8708—Lighting.

This account covers the cost of the material below and the labor of its installation.

14 carbon lamps, 16 c.p.	260 ft. brewery cord
3 tungstens, 250 watt	300 ft. No. 12 wire
100 ft. conduit.	

MACHINE AND CARPENTER SHOP

Account 8715—Excavation.

Same as account 8701.

Account 8716—Foundation.

Same as account 8702.

Account 8717—Steel Structure.

This building is the same as 8703 account. There were used here 38.23 tons of structural steel. (See Fig. 105.)

Account 8717.1—Doors, Windows and Frames.

This account is the same as 8703.10, with the following list of material:

Thirteen 11 ft. 7 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 63 lights, 2 mullions, with 3–6 light ventilators, unglazed.
Two 10 ft. 3 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 56 lights, 1 mullion, no ventilators, unglazed.
Two 10 ft. 3 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 48 lights, 1 mullion, no ventilators, unglazed.
Two 4 ft. by 9 ft. steel sliding doors, with six 14 in. by 20 in. lights, unglazed, lower panels steel.
One 11 ft. by 12 ft. steel sliding door, with forty 14 in. by 20 in. lights, unglazed, lower panels steel.
Two 14 ft. by 20 ft. Kinnar steel rolling doors.

Fourteen 14-ft. 10-in. lintels, made of 8-in. channels.
Six 11-ft. 6-in. lintels, made of 8-in. channels.
Seven hundred forty 14 in. by 20 in., ¼ in. thick factory ribbed glass window panes.
One hundred sixty-four 13½ in. by 19½ in., ¼ in. thick factory ribbed glass window panes.
Eighty-two 14 in. by 19½ in., ¼ in. thick factory ribbed glass window panes.
Twenty 15 in. by 20 in., ¼ in. thick factory ribbed glass window panes.

Steel window and door frames for the above list. (See Fig. 106.)

Account 8717.11—Concrete Sills.

Same as account 8703.11, but the following product:

Thirteen 8½ in. by 10 in., 14 ft. long sills.
Two 8½ in. by 10 in., 11 ft. 6 in. long sills.
Two 8½ in. by 10 in., 6 ft. 6 in. long sills.
(See Fig. 21.)

Account 8717.20—Tile Walls.

Same as account 8703.20.

Account 8717.21—Unloading Tile.

Same as account 8703.21.

Account 8717.22—Wall Coping.

Same as account 8703.22.

Account 8717.30—Roof.

Same as account 8703.30. This roof contains 77.21 squares, equal to 14,543 b.m. 2 by 8 lumber.

Account 8717.31—Ventilators.

Same as account 8703.31. Three 48-in. Burt ventilators used here.

Account 8717.40—Floor.

This account covers the cost of the material and labor required to lay this floor. Six inch by eight inch stringers were laid 2 ft. 6 in. on centers with earth tamped in between them. On the stringers No. 3 grade, 3 in. by 12 in. white cedar planking of various lengths was spiked down. (See Fig. 24.)

Account 8717.50—Benches.

Same as account 8703.50.

Account 8717.60—Painting.

Same as account 8703.60.

Account 8718—Crane.

This is the same as account 8704, with the exception that the crane here used is of 5 ton capacity.

Account 8719—Tools.

This account covers the purchase price of all the material listed below and the labor cost of installing it:

	Factory	Freight	Clifton
1 Prentiss machine bench vise, No. 21.	.....	.....	$20.15
1 machine bench vise, No. 21.....	.....	.....	20.16
1 machine bench vise, No. 22.....	.....	.....	28.85
1 machine pipe vise, No. 2A.....	.....	.....	2.38
1 machine pipe vise, No. 4A.....	.....	.....	7.77
1 stationary bench vise, No. 56.....	.....	.....	20.72
40 ft. of 1½-in. pipe.....	.....	.....	2.97
1 No. 48 power grindstone.....	.....	.....	56.62
2 emery wheels.....	.....	.....	8.90
1 emery wheel grinder.....	.....	.....	17.00
1 No. 40 special turning machine.....	.....	.....	36.22
1 set faces for wiring machine.....	.....	.....	5.56
1 gauge.....	.....	.....	2.35
1 burr machine and stand.....	.....	.....	9.92
1 No. 17 S. P. crimper and stand.....	.....	.....	10.77
1 No. 3 beading machine.....	.....	.....	26.79
1 No. 0236 squaring shears.....	.....	.....	180.86
1 stake-holder and stakes.....	.....	.....	42.15
1 rivet set.....	.....	.....	2.65
1 No. 101 tinner's rule.....	.....	.....	2.73
1 power hack saw No. 3.....	.....	.....	29.63
1 radial drill press, 42 in.....	.....	.....	752.20
Miscellaneous.....	.....	.....	21.92
1 50-in. cornice brake.....	.....	.....	155.96
1 16-in. rip saw.....	.....	.....	4.30
Castings.....	.....	.....	10.10
1 No. 1 drill chuck.....	.....	.....	5.61
1 No. 2½ drill chuck	.....	.....	7.02
72 hack saw blades.....	.....	.....	5.55
1 surfacer, 20 in. by 6 in.....	$180.00	$26.70	206.70
1 No. 50 band saw.....	175.00	27.45	202.45
1 lathe, 14 in. by 8 ft.....	563.75	81.40	645.15
1 lathe, McCabe patented double spindle.	2,111.00	277.15	2,388.15
1 Crescent saw table.....	168.75	51.34	220.09
One 20-in. Rockford shaper.....	425.00	175.07	600.07
One 2-in. bolt cutter.....	355.00	47.10	402.10
1 Crane pipe machine 2 in.....	192.00	16.56	208.56
1 Crane pipe machine 4 in.....	480.00	44.10	524.10
1 Crane pipe machine 12 in.....	1,500.00	163.59	1,663.59
Small tools, miscellaneous equipment	.....	.....	394.36
			$8,953.13

Account 8720—Shafting, Pulleys and Belting.

This account covers the purchase price of the material below and its cost of installation:

1 pc. 30 ft., 2 11/16 in. diameter shafting.
1 pc. 60 ft., 2 11/16 in. diameter shafting.
1 pc. 18 ft., 2 11/16 in. diameter shafting.
1 pc. 18 ft., 2 7/16 in. diameter shafting.
1 pc. 22 ft., 2 7/16 in. diameter shafting.
1 pc. 10 ft., 2 7/16 in. diameter shafting.
1 pc. 4 ft., 2 7/16 in. diameter shafting.

Many pulleys ranging from 10 in. to 68 in. diameter, with necessary hangers, collars, boxes, etc. (See Fig. 19.)

Account 8721—Motor.

This account covers the purchase price and cost of installing the following motor.

One 40-h.p., 440-volt, 3-phase, 60-cycle, 850-r.p.m. motor.

(See Fig. 20.)

Account 8722—Lighting.

This covers the cost of the following material and the labor of installing it.

17 carbon lamps	16 c.p.
3 tungstens	250 watt
240 ft. brewery cord	
100 ft. conduit	
360 ft. No. 12 weatherproof wire.	

(See Fig. 23.)

GENERAL OFFICE

Account 8804—Furniture and Fixtures.

This account covers the furniture and fixtures purchased for the smelter office, which to date has not been built. The furniture is in use in the temporary offices.

WAREHOUSE

Account 8810—Excavation.

This was the same as 8701.

Account 8811—Foundation.

This was the same as 8702, save that the walls were higher.

Account 8812—Steel Structure.

This building is the same type as the boiler and machine shops, save a corrugated iron roof was used in place of a wood and paper covering. There were 26.5 tons of structural steel used and 13.26 tons of corrugated iron. (See Fig. 26.)

Account 8812.1—Doors, Windows and Frames.

The doors for the warehouse were similar to the boiler and machine shops. The lintels over the windows and doors were the same as in

the shops. The small doors, all windows and frames were wood. This account covers the cost of the door and window material listed below and the labor of installing the same.

51 windows, 3 ft. 9½ in. by 7 ft. 8¾ in. by 1¾ in. 48 of these were grouped in triple frames, all glazed	$305.99
16 wood frames for 48 of above windows	170.32
Lumber for 3 window frames, all door frames and all hardware	83.78
1 O. G. 1 light glazed door, 3 ft. 6 in. by 7 ft. 1 3/8 in.	7.91
Two 9 ft. 10 in. by 7 ft. 6½ in. Kinnar rolling doors	157.30
Steel lintels	331.01
(See Fig. 27.)	
	$1,056.31

Account 8812.11—Concrete Sills.

See account 8703.11. Sills were made here for frames of account 8812.10. (See Fig. 22.)

Account 8812.2—Tile Walls.

Same as for account 8703.11.

Account 8812.21—Unloading Tile.

Same as for account 8703.21. (See Fig. 26.)

Account 8812.22—Coping.

Same as for account 8703.22. (See Fig. 28.)

Account 8812.3—Painting Roof.

This covers the labor and material of painting underside of corrugated iron roof 2 coats of lead and linseed oil, cream color.

Account 8812.31—Ventilators.

Same as for account 8703.31. These three ventilators were 48 in. diameter with round base. (See Fig. 25.)

Account 8812.40—Floor Excavation.

This entailed cutting down the front in the warehouse 6 to 8 in. and backfilling in places.

Account 8812.41—Floor Concrete.

This concrete floor was cast in large 6 ft. to 8 ft. blocks, 4 in. thick, with sand joints between blocks. The concrete was hand mixed in the proportions of 6 sand and gravel to 1 cement. It was transported in wheelbarrows 100 ft. The top finish, ¾ in. thick, was 2 sand to 1 cement. This top was troweled smooth.

Account 8812.50—Lighting.

This account covers the cost of the following material and the labor of installation.

26 carbon lamps, 16 c.p. 365 ft. brewery cord
170 ft. No. 12 weatherproof wire.
(See Fig. 29.)

Account 8813—Fixtures.

This account covers the purchase price of the steel bins, shelving, counter scales, office partition and furniture, as listed below; also the erection cost.

197 ft. Berger sectional steel bins and shelving. See sketch 30. Lineal feet refers to half of bins shown by sectional elevation. Bins received knocked down, gauge of material 16 to 20.....	$1,116.82
1 No. 1046 dormant warehouse scales, weighing 5,000 lb. to ½ lb.....	141.42
Furniture, material for office partition, etc.....	282.88
	$1,541.12

Account 8813.10—Painting.

This account covers the cost of material and labor of a lot of miscellaneous painting at the warehouse. The steel doors were given one coat of turkey red. The iron lintels were given one coat of lamp black in linseed oil. The counter was stained and oiled.

Account 8813.11—Painting Sash.

This account covers the cost of material and labor used to paint all the warehouse sash. They were given two coats of white lead and linseed oil, cream color.

LABORATORY

Account 8820—Excavation.

This covers the excavating for the laboratory walls and basement in red clay with boulders and gravel. It was done with pick and shovel, and wheeled 75 ft. in barrows. Some backfilling for the floors in 3-in. layers is also here included.

Account 8821—Foundation.

This covers the concrete building walls which were machine mixed, in the proportions 8 sand and gravel to 1 cement. The concrete was handled in wagons 250 yd. The walls were 12 in. at top to 18 in. at bottom. One hundred per cent. of the vertical surface was formed.

Account 8821.1—Plain Concrete Floors.

These floors were mixed, 5 sand and gravel to 1 cement, in a machine, transported 1,000 ft. in wagon and laid 4 in. thick with a smooth finish. Sand joints were used. The top finish was ¾ in. thick, 2 parts sand to 1 cement, and was troweled smooth.

Account 8821.2—Reinforced Floors.

These floors were formed, two way reinforced with ½-in. and ⅝-in. rods. In other respects they were the same as 8821.1.

Account 8821.3—Sills and Lintels.

The sills and lintels used at the laboratory were separately molded reinforced concrete. The concrete was made 3 parts sand and gravel to 1 cement. Three $\frac{5}{8}$-in. rods ran the entire length of both sills and lintels. The lintels were 8 in. by 8 in. by 5 ft. The sills were $4\frac{1}{2}$ in. by 9 in. by 4 ft. 1 in. (See Fig. 34.)

Account 8822—Tile Walls.

This account covers the tile, mortar, scaffolds, mason labor, carpenter labor, and hauling incident to building the tile walls of the laboratory.

Account 8822.2—Carpenter Work.

This account covers the material and carpenter labor incident to the installation of the partitions, ceilings, and roof structure. (See Fig. 31.)

Account 8822.5—Doors, Windows and Frames.

This account covers the cost of the following material and the labor of installing the same, together with the necessary frames.

16 windows 6 ft. 8 in. by 3 ft. $1\frac{1}{4}$ in. by $1\frac{1}{4}$ in. glazed.
5 sash with four, 12 in. by 24 in. lights, glazed.
1 sky light 6 ft. $3\frac{1}{16}$ in. by 6 ft. $9\frac{1}{4}$ in. with 36, 12 in. by 14 in. double strength glass
1 sky light 6 ft. $3\frac{1}{16}$ in. by 4 ft. $6\frac{1}{2}$ in. with 24, 12 in. by 14 in. double strength glass.
1 sky light 4 ft. $1\frac{1}{4}$ in. by 4 ft. $6\frac{1}{2}$ in. with sixteen 12 in. by 14 in. double strength glass.
4 doors 2 ft. 8 in. by 6 ft. 8 in. by $1\frac{1}{4}$ in., glazed.
4 doors 2 ft. 8 in. by 6 ft. 8 in. by $1\frac{1}{4}$ in. glazed.

(See Figs. 32 and 33.)

Account 8824—Wood Fixtures.

The account covers the cost of the laboratory hoods, stacks, benches, cabinets and the like.

Account 8825—Lighting.

This account covers the wiring in the laboratory for lights, hot plates, furnaces, etc.

Account 8826—Plumbing.

This account covers the purchase price of the material below and the labor of installing it.

1 flush closet — 1 distilling apparatus
2 sinks and drains — 1 water tap
Piping, fittings, lead, etc.

Account 8828—Painting.

This covers the cost of the material and labor incident to painting at the laboratory. It is of no value for unit costs.

Account 8829—Plastering.

This covers the cost of plastering material, scaffolds, plasterer and carpenter labor used in plastering the inside tile walls of the laboratory. The plastering was contracted at 18 cents a square yard, while the company furnished all material and carpenter labor.

Account 8830—Apparatus.

This account covers the purchase price of the material segregated below, and the labor of setting up the same.

Five 12 in. by 18 in. by 6 in. 110-volt hot plates	$143.08
1 Thompson's analytical balance, style 28	91.65
1 distilling apparatus	62.70
1 Hoskins electric furnace	225.53
1 electric drying oven	41.79
Brushes, tubing, funnels, etc	54.23
	$618.98

Account 8831—Oil Centrifuge.

This apparatus was located near the oil tanks in a corrugated iron shed, 4 ft. by 4 ft. by 8 ft. The labor in this account was for the building as well as setting up the centrifuge and delivering the material.

1 Braun oil centrifuge, vertical, direct compound type, 110 volt, alternating	$139.25
Material for shed	18.34
	$157.59

SAMPLE ROOM

Account 8841—Excavation.

This excavation covered the making of a thin top slice and shaping the ground for a plain concrete floor. The dirt was red clay. It was done with pick and shovel and cast to one side.

Account 8842—Foundation.

This covers the making of a few small concrete walls, machine mixed, 8 sand and gravel to 1 cement. Seventy-five per cent. of the vertical surface was formed. The concrete was handled about 450 ft.

Account 8842.1—Concrete Floors.

These floors were of smooth troweled concrete, 5 in. thick, machine mixed, 5 sand and gravel to 1 cement, the top finish 1 in. thick, 2 sand and 1 cement. All of the material was hauled in wagons 500 ft. The concrete was laid in blocks with sand joints.

Account 8843—Walls and Roof Structure.

This account covers the purchase price of the following material and the cost of erection.

Lumber for sides and roof rafters	$148.12
Corrugated iron	47.68
Nails, etc.	6.01
	$201.81

(See Fig. 35.)

Account 8843.4—Roof.

This account covers the roof sheathing, composition paper, and the labor incident to installing it.

1,008 b.f., 1 in. by 12 Oregon pine	$31.50
10 squares 3-ply asbestos roofing	45.07
	$76.57

Account 8843.5—Doors and Windows.

This account covers the cost of the following material, and the cost of installing the same:

Two 3 ft. by 7 ft. by 1⅜ in. doors Twelve 2 ft. 11¼ in. by 5 ft. 3⅝ in. by 1⅜ in. windows, glazed Frames for doors and windows	$115.24
Miscellaneous	3.61
	$118.85

Account 8844—Oven.

This account covers the making of a drying oven, together with the cost of material and installation. The oven was made in the shops of $\frac{3}{16}$-in. sheet plate, 4 ft. by 2 ft. by 6 ft. high, and lined with $\frac{3}{16}$-in. asbestos mill board. It had suitable shelves of pipe coils.

One oven	$46.77
Steam piping, etc	11.79
	$58.56

Account 8845—Motor Platforms and Fixtures.

This covered the cost of the following material and its installation:

4 bucking boards, 3 ft. diameter by 2¼ in. thick	$61.80
21 sheets galvanized iron (No. 18) for pans, etc	53.06
Lumber for one bench top 3 ft. by 12 ft. for pulverizers, and one bench top 2 ft. by 12 ft. for cutting samples	14.74
	$129.60

Account 8846—Lighting.

This covers the cost of the lighting wiring for seven lights.

Account 8848—Painting.

This covers the cost of material and labor incident to painting the sash of this building, two coats of white lead and linseed oil.

Account 8849—Machinery.

This covers the purchase price of the list of material given below, together with the labor of installing it.

	Factory	Freight	Clifton
Two 2 by 6 roll jaw crushers. (Sturtevant Milling Co.)	$250.00	$51.80	$301.80
4 jaw plates	10.00	0.80	10.80
4 shields	3.00	0.22	3.22
8 toggles	3.00	0.14	3.14
4 toggle seats	4.00	0.11	4.11
2 Jones samplers 10 by 18 in	40.00	2.25	42.25
4 Braun pulverizers, 9 in. pulley	.....	.....	363.48
2 Braun coal grinders, tight and loose pulleys, 12 in. diameter	.....	.....	147.62
Moisture scales	.....	.....	20.11
Sieves and miscellaneous	.....	.....	15.47
			$912.00

Account 8849.1—Motor.

This covers the purchase price and cost of installing the following motor.

One 5-h.p. Westinghouse motor $116.77

Account 8849.2—Shafting, Pulleys and Belting.

This account covers the cost of the material segregated below and the labor of installing it:

One 24 in. by 6 in. by 1 7/16 in. bore wood split pulley Four 26 in. by 6 in. by 1 7/16 in. bore wood split pulleys Two 22 in. by 4 in. Princeton clutch pulleys 26 ft. 1 in. of 1 7/16 in. shafting 4 rigid ring oiling pillow blocks Two 1 7/16 in. safety set collars	$138.13
Leather belting, 2 in., 3 in. and 5 in.	32.04
	$170.17

MISCELLANEOUS ACCOUNTS

Account 8901—Derricks and Construction Equipment.

This account is indirect expense, see 8999. The charge covers the cost of all derricks, concrete mixers, carts, wagons, picks, shovels, in fact every tool used during construction.

Account 8902—Sewer System—Cost of Pipe and Laying.

This account covers the cost of the material and labor of laying the same used in the sewer system. The sewer lines aggregated 2,967 ft. of vitrified sewer pipe, ranging from 6 in. to 15 in. in diameter. It was all laid an average depth of 4 ft. below the surface. Concrete manholes for the system are in account 8902.2.

Account 8902.1—Sewer System—Excavation.

This excavation covers the entire trenching or tunneling as was in some cases done for the sewer system. All kinds of soil were run through. The trenches varied from 18 in. to 60 in. wide, and from 2 ft. to 20 ft. in depth.

Account 8902.2—Sewer System—Concrete.

This concrete covered a large number of small jobs including manholes and the like along the sewer lines. In general the mix was 7 sand and gravel to 1 cement.

Account 8903—Permanent Outside Closets.

This covers 3 latrines built of 2 by 4's, 4 by 4's and corrugated iron. In size they were 8 ft. by 19 ft. 6 in. by 8 ft. high. The closet was built of concrete and is cleaned by a periodical flushing of water which is siphoned into a sewer.

Account 8904—Telephone System.

This account is indirect expense, see 8999. It covers the cost of a telephone line from Clifton to the smelter, together with all phones and connections about the plant during construction.

Account 8905—Permanent Outside Lighting.

The material account here is as follows:

Four 110 volt flaming arc lamps	$142.40
Conduit, wire, etc	35.59
	$177.99

The poles here used were old ones previously charged out to construction equipment. The lights were placed at various needed points about the plant and then connected to the nearest 110-volt circuit. Labor costs cover the setting of poles, the connecting of lamps, running a conduit line down each pole to a switch box, and installing the switch box.

Account 8905.1—Temporary Outside Lighting.

This account is indirect expense, see 8999.

Account 8906—Water Pipe Lines—Excavation.

This excavation covers the trenching for all the water lines. It represents all sorts of material excavated from 8 ft. to 15 ft. in depth.

Account 8906.01—Water Pipe Lines—Concrete.

This was a small amount of concrete used to anchor the 6-in. line at the foot of the hill, as it comes down from the tank.

Account 8906.02—Water Pipe Lines—Cost and Laying.

This account covers the cost of the material and labor of laying all the water lines about the smelter.

There was 2,052 ft. of 6-in. pipe
1,058 ft. of 4-in. pipe
200 ft. of 2½-in. pipe
268 ft. of 2-in. pipe
115 ft. of 1½-in. pipe
50 ft. of 1-in. pipe

4,253 ft.

Total of 4,253 ft. with all necessary fittings, valves, and fire hydrants.

Account 8906.1—Six-Inch Pipe Line from Clifton.

This account covers the cost of the material and labor incident to laying a 6-in. water pipe line, 8,988 ft. long, from Clifton to smelter. It includes excavating, painting, and backfill.

Account 8906.2—Water Supply Tank—Excavation.

This covers the making of a 3-ft. slice for a water tank foundation. It involved the use of powder, and was handled with picks, shovels, and wheelbarrows.

Account 8906.4—Water Supply Tank—Cost and Erection.

This tank was erected on a hill at an elevation of 200 ft. above the tracks where the material was received. It is 40 ft. in diameter, 26 ft. 9½ in. high, and has a steel cover. Its capacity is 250,000 gallons. The tank steel was erected by contract. The account stands thus:

Road work	$72.80
Hauling	121.31
33.67 tons, tank erected	3,550.00
Freight on tank	235.69
Power for riveting	92.31
Miscellaneous	64.92
	$4,137.03

Account 8907—Watchman.

This account is indirect expense, see 8999. It covers the pay of watchman during the construction period.

Account 8908—Power Distribution.

This account covers the cost of material and labor incident to the following work. Two- and three-inch conduits were run underground

from the power house to the heater house and to the converter building. Along these lines four concrete manholes or pull boxes were installed At the converter building the wires from the conduits run up a steel column and from thence on steel brackets through the reverberatory and boiler building to a point of distribution by poles. From this point the wires go to the roasters, sample mill, crushing plant, and bedding plant. The wire ranged from 1,000,000 c.m. to No. 12 weatherproof.

Account **8908.1—Temporary Oil Tanks.**

This account is indirect expense, see 8999. It covers the cost and erection of an oil tank with pipe lines used during the construction period.

Account **8909—Permanent Air Line—Excavation.**

This covers the cost of trenching and backfilling for the air lines. The trenches were in every kind of soil, 18 in. to 6 ft. deep and 1 ft. to 3 ft. wide.

Account **8909.1—Permanent Air Line—Laying.**

This account covers the cost of material used and the labor of installing the same. The lines together are 2,316 ft. long and composed of the following quantities of different sized pipe.

526 ft., 1-in. pipe	656 ft., 2-in. pipe
36 ft., 1½-in. pipe	838 ft., 3-in. pipe
80 ft., 1½-in. pipe	180 ft., 4-in. pipe

Account **8910—Transmission of Power to Various Departments.**

This is a suspense account which has been charged away.

Account **8911—Watchman House.**

This account is indirect expense, see 8999. It covers the cost of a temporary building.

Account **8912—Tool Shed.**

This account is indirect expense, see 8999. It covers the cost of a temporary building.

Account **8913—Barn and Corral.**

This account is indirect expense, see 8999. It covers the cost of the corral and barn.

Account **8914—Temporary Blacksmith Shop.**

This account is indirect expense, see 8999. This is a temporary building used during construction.

Account **8916—Temporary Power Plant.**

This account is indirect expense, see 8999. It covers a large compressor, Maune type boiler and temporary building, together with the labor of installation.

Account 8917—Temporary Crushing Plant.
This account is indirect expense, see 8999. It covers the cost and installation of a 10-in. by 20-in. crusher, 65-ft. bucket elevator, rock and sand bins, shafting, belts, pulleys and motors.

Account 8918—Temporary Water Tanks.
This account is indirect expense, see account 8999. It covers several small tanks erected for use at the temporary camp site, as well as a large general supply, wooden tank.

Account 8919—Temporary Electrical Shop Equipment.
This account is indirect expense, see 8999. It covers a temporary shop equipment.

Account 8920—Wagon Roads.
This account is indirect expense. It covers the building of many roads and trails required during the construction period.

Account 8921—Temporary Pumping Plant.
This account is indirect expense. It covers a timbered shaft, pump and pipe line.

Account 8922—Temporary Pipe Lines.
This account is indirect expense. It covers the cost of all temporary water and air lines laid during the construction period.

Account 8923—Temporary Warehouse.
This account is indirect expense. It covers the cost of the temporary warehouse and equipment.

Account 8924—Temporary Cement Sheds.
This account is indirect expense, see 8999. It covers the cost of corrugated iron sheds for storing cement.

Account 8925—Horses, Harness and Carts.
This is indirect expense, see 8999. This covers the cost of some carts, work harness, and the purchase price of an animal killed during construction.

Account 8926—Temporary Office.
This account is indirect expense, see 8999. This covers the cost of the construction office.

Account 8927—Temporary Lavatories.
This account is indirect expense, see 8999.

Account 8928—Temporary Machine Shop.
This account is indirect expense, see 8999. This covers the cost of temporary machine shop.

Account 8929—Employees' Railroad Transportation.
This account is indirect expense, see 8999. During the construction period 300 to 500 men were employed, over three-fourths of whom lived in and around Clifton. The cost of transporting these men back and forth each day was borne by the construction.

Account 8930—Clearing Land.
This account is indirect expense, see 8999.

Account 8931—Test Holes.
This account is indirect expense, see 8999.

Account 8933—Furniture and Fixtures.
This account is indirect expense, see 8999. It covers the temporary office fixtures.

Account 8934—Miscellaneous Supplies.
This account is indirect expense, see 8999.

Account 8935—Shop Equipment.
This account is indirect expense, see 8999. It covers material used in various temporary shops.

Account 8936—Overhead Shop Expense.
This account is indirect expense, see 8999. This account covered all labor and supplies used in the shops not directly charged to the work in hand.

Account 8937—Stock Lumber.
This is a suspense account.

Account 8938—Powder Magazine.
This account is indirect expense, see 8999.

Account 8939—Miscellaneous Labor.
This is indirect expense, see 8999.

Account 8941—Temporary Railway Receiving Bins.
This account is indirect expense, see 8999.

Account 8942—Water Supply.
This account is indirect expense, see 8999. It covers the cost of water from Clifton used during construction.

Account 8943—Corral Expense.
This was suspense account distributed at the close of each month, on the basis of the cost per animal-day.

Account 8944—Switching and Freight from Clifton.
This account is indirect expense, see 8999. It was impossible to place these charges to the material freighted, owing to incomplete records.

Account 8945—Office Stationery and Supplies.
This account is indirect expense, see 8999.

Account 8946—Warehouse Operating Expense.
This account is indirect expense, see 8999. It covers the expense of running the warehouse during the construction period.

Account 8927—Timekeeping Expense.
This account is indirect expense, see 8999. It covers the expense of the timekeeping and distribution during the construction period.

Account 8948—Form Lumber.
This is a suspense account. Its money was apportioned to all concrete costs on the basis of board-feet of lumber used on each job.

Account 8949—Cement.
This was a suspense account. It was apportioned to all concrete accounts on the basis of sacks used.

Account 8951—Sand and Gravel.
This is a supense account. Its money was apportioned to the different concrete accounts on the basis of cubic yards of concrete cast.

Account 8952—Employees' Quarters.
This account is indirect expense, see 8999. It represents the cost of a boarding house, bath house, and 42 tent houses, less the rent they paid on during the construction period.

Account 8953—Crushing Plant Operating Expense.
This is a suspense account. Its money was apportioned to the different concrete jobs on the basis of cubic yards of concrete benefiting.

Account 8954—Concrete Power and Repairs.
This is a suspense account, see 8953.

Account 8955—Mortar Sand.
This is a suspense account. It was apportioned to the brick-work on the basis of the amount used by the various jobs.

Account 8955.1—Mortar Lime.
This is a suspense account, see 8953.

Account 8955.2—Mortar Cement.
This is a suspense account, see 8955.

Account 8955.4—Fire Brick Mortar.
This is a suspense account, see 8955.

Account 8955.5—Silica Brick Mortar.
This is a suspense account, see 8955.

Account 8956—Operating Temporary Power House.

This is a suspense account. It was apportioned to the various jobs on the basis of horse power days.

Account 8957—Maintenance of Track in Yards.

This account is indirect expense, see 8999. It represents the upkeep of tracks during construction, and the cost of many temporary construction tracks.

Account 8958—Ditch at Tunnel No. 2.

This account is indirect expense, see 8999. It represents a ditch built by the Arizona & New Mexico Railway, to protect their main line which has been endangered by surface water as a result of the site chosen for the smelter.

Account 8961—Steam Heating System—Excavation.

This covers the making of a long shallow trench and backfilling it. The ground was red clay.

Account 8961.1—Steam Heating System—Cost and Installation.

This covers the cost of laying 260 ft. of 2-in. steam pipe and 236 ft. of 2½-in. steam pipe in a 2-in. lumber box. The pipe was covered with double standard magnesia covering.

Account 8975—Cleaning Up.

This account is an indirect charge, see 8999. It represents cleaning up about the plant after construction.

Account 8976—Rehandling Brick and Tile.

This account is an indirect charge, see 8999. It represents handling tile not directly chargeable to the jobs where the tile was used, but as a result of other considerations.

Account 8998—Direct Charges.

This account is indirect expense, see 8999. It represents direct charges to indirect expense.

Account 8999—Indirect Expense.

This account is a summation of the accounts, as listed under charges to indirect expense. As the total of indirect expense, they represent a percentage of the total cost of the smelter, less the engineering and indirect expenses, and have been so reported. In the making of any total estimate based on the unit costs derived from this sheet, it is assumed that of the total estimate, 7.53 per cent. will be taken to ascertain the item of indirect expense.

POWER PLANT

Power House

Account 9001—Excavation.

This excavation was a large cut, about 55 ft. by 280 ft. by 10 ft., for the basement of the power house, the machine foundation and the building piers. The material encountered was red clay and boulders on top, with sand and gravel beneath which was saved for concrete material. Powder was used, followed by plowing, picks, shovels, fresnos, and carts. The material was hauled 450 ft. (See Fig. 53.)

Account 9002—Building Foundation Piers.

This concrete was cast in piers which supported the steel columns. The piers were about 3 ft. by 4 ft. by 3 ft. plain concrete, hand mixed, in the proportions of 7 sand and gravel to 1 cement, and transported 150 ft. in wheelbarrows to place. A great deal of difficulty was experienced here in keeping out the sand and gravel which constantly sloughed in from the sides. Fifty per cent. of the vertical surface was formed.

Account 9002.1—Building Foundation Walls.

This concrete was cast as a long reinforced wall running around outside and bearing against the columns of the building. It is 12 in. at top, 20 in. at bottom, 11 ft. high, designed as a slab to withstand earth pressure on a 20-ft. span. The concrete was machine mixed in the proportions of 5 sand and gravel to 1 cement, wheeled 150 ft. to place and reinforced with $\frac{7}{8}$- and $\frac{3}{4}$-in. rods variously spaced. Ninety-five per cent. of the vertical surface was formed. (See Fig. 47.)

Account 9002.2—North Tunnel.

This concrete was cast as a box culvert in an open cut. It is about 6 ft. by 7 ft. by 300 ft. long. It was hand mixed in the proportions of 5 sand and gravel to 1 cement, wheeled 100 ft. to place and reinforced with $\frac{1}{4}$ in. and $\frac{1}{2}$-in. rods. Fifty-five per cent. of the vertical surface was formed, as well as the roof of the culvert. (See Fig. 50.)

Account 9002.3—Concrete Drain.

This concrete forms a drain through the middle of the power-house basement. There are two walls, 10 in. thick, 18 in. high, 3 ft. apart, with a 9-in. bottom in the drain laid on grade. The concrete was one-half machine and one-half hand mixed, in the proportions of 7 sand and gravel to 1 cement, wheeled in barrows 150 ft. to place. Ninety per cent. of the vertical surface was formed.

Account 9002.4—Basement Floor—Concrete.

This concrete was cast as a floor on grade in the basement of the power house. It was laid in 6-ft. blocks with sand joints. The mixture,

machine made, was 7 parts sand and gravel to 1 cement. The top finish was ¾ in. thick and made 2 parts sand to 1 cement. The total thickness of the floor is 4 in. The concrete was transported in barrows on an average of 125 ft.

Account 9002.45—Basement Floor—Painting.

This account covers material cost and labor of cleaning floor and applying two coats of Toch Brothers cement filler.

Account 9002.6—Preparing of Concrete for Painting.

This covers the cost of preparing basement concrete walls for painting. The air holes were filled, rough surfaces smoothed, and one coat of cement grout applied.

Account 9002.7—Painting Concrete.

This account covers the cost of the material and labor of applying it to the power house basement walls. One coat of Wadsworth Howland Bay State cement coating was put on. This cost $1.88 a gallon f.o.b. Clifton. The material account is as follows:

150 gallons of paint	$281.40
Brushes	8.42
Turpentine	5.71
Miscellaneous	6.08
	$301.61

Account 9003—Steel Structure.

(See account 7308.2)

This account covers all the structural steel including railings and the like used in the power house building. There were 254.29 tons. (See Fig. 54.)

Account 9003.1—Tile Walls.

This account covers the cost of all the tile, mortar material, scaffolds, mason labor, carpenter labor, and hoisting power required to lay up the walls. The walls were 40 ft. high on the average from the ground level. The walls were 8 in. thick, laid directly upon the concrete walls. (See account 9002.1.) Two mortars were used in laying up the tile, one with lamp black in to give the outside joint a pleasing appearance. The joints were struck. The mortar mixture was 1 cement, 3 sand, and 1 lime. (See Figs. 52 and 54.)

Account 9003.11—Unloading Tile.

This account covers the preparing of the site for unloading, checking quantities and unloading the tile used in the power plant.

Account 9003.12—Wall Coping.

Similar to account 8703.22, which see. The depth of this coping was, however, 18 in. against 12 in. at the warehouse and shops.

Account 9003.2—Doors, Windows and Frames.

This account covers the doors, sash, hardware and frames listed below, together with the labor cost of installing the same. The steel frames were made by the Kansas City Structural Steel Co.

	Factory	Freight	Clifton
105 single sash, 2 ft. 8¾ in. by 5 ft. 1½ in. by 1¾ in. glazed with 15 ribbed glass lights, fixed sash	$325.50	.....	.....
1 single sash, 2 ft. 8¾ in. by 4 ft. ½ in. by 1¾ in. glazed with 12 ribbed glass lights, fixed sash	26.00	.....	.....
73 windows, 2 ft. 8¾ in. by 6 ft. 3 in. by 1¾ in. glazed with 18 ribbed glass lights, double hung	284.70	.....	.....
40 sash, 3 ft. ¾ in. by 2 ft. 6½ in. by 1¾ in glazed with 6 ribbed glass lights, hinged.	56.00	———	———
		$93.05	$785.25
Steel frames for all but the 40 6-light sash and 6 door frames, 3 ft. 1 in. by 7 ft. 6 in.	1,815.20	292.77	2,107.97
300 Caldwell sash balances, No. 18	.....	.....	111.95
3,884 ft. b.m. lumber for frames for 40 single sash.	.....	.....	91.44
Locks, hinges, nails, etc	.....	.....	223.32
			$3,319.93

(See Fig. 51.)

Account 9003.21—Concrete Sills.

This account covers the material used and the labor expended in making the power house concrete sills. About one-half of the sills were cast in place and the remainder as separately molded members. They were 8 in. by 10 in. for the windows and 3 in. by 10 in. for the fixed sash at the top of the building. The material was hand mixed, 3 sand to 1 cement, and 3 ⅝-in. rods ran the length of the sills. (See Fig. 51.)

Account 9003.3—Ventilators.

This covers the cost of 6 '48-in. Burt ventilators with square base and the labor of installing them on the roof of the power house. They were lifted onto the roof with a locomotive crane.

6 Burt ventilators	$280.17	$152.57	$432.74
Miscellaneous			7.02
			$439.76

Account 9003.4—Main Floor Columns.

This account covers the cost of the material, fabrication, and labor of erecting 57 pipe columns and 11 structural steel columns to support the steel beams of the power house floor. The 57 pipe columns were made of 4-in. pipe, with a cast-iron base and capital, and were so placed

that they might be easily shifted a foot or two along the low flange of the I beams they support, should they interfere with future piping schemes. The structural columns were permanently located. The material segregated is as follows:

Caps and bases, cast iron	$227.80
668 ft. of 4-in. pipe	186.40
Structural steel and rivets	136.99
Miscellaneous	75.25
	$626.44

Account 9003.41—Main Floor Slab—Concrete.

This account covers the cost of the Berger multiplex plate laid on the steel I beams of the power-house floor, and covered with concrete reinforced with ½-in. and ¼-in. rods. The Berger plate was laid upon steel beams and not wired; ¼-in. and ½-in. rods, 1 ft. on centers, were laid in the concrete at right angles with the grooves of the plate. The concrete mixture was 7 parts sand and gravel to 1 cement, machine made. The top finish was proportioned 2 sand to 1 cement. It was laid ¾ in. thick, smooth troweled, and marked off in 6-ft. squares. The material account stands thus:

102.1 squares, 3-in. Berger plate	$2,101.88
¼-in. and ½-in. reinforcing steel	62.06
Concrete materials	1,177.67
	$3,341.61

(See Fig. 55.)

Account 9003.42—Painting Under Side Main Floor.

The Berger plate exposed beneath the floor of 9003.41 was cleaned of rust and painted two coats of linseed oil and white lead, cream color. This necessitated low scaffolds. The square yards in which the unit cost is reported are the yard measurements derived from developing the plate. This cost covers the material and the labor.

Account 9003.43—Painting Top Main Floor.

This account covers the labor and material used in painting the upper surface of the power house Berger plate concrete floor. It was given two coats of Toch Brothers cement filler and one coat Toch Brothers warm gray cement paint, after the floor had been well cleaned and dried out.

Account 9003.5—Berger Multiplex Plate.

This account covers the material and labor used in putting Berger plate upon the power-house roof. The eaves of the roof are 41 ft. from the ground and the roof is ¼ pitch. The Berger plate was hoisted

to place by use of a single pulley and hand rope. When laid in position it was wired to the purlins with No. 10 wire. The segregated account stands as follows:

Berger plate	$2,962.25
Wire	20.41
Tools, etc	80.52
	$3,063.18

(See Fig. 56.)

Account 9003.51—Roof—Concrete.

This account covers the concrete placed on the roof. It was hoisted to the eaves at various places and transported to position in hand buckets. The concrete was hand mixed, 5 sand and gravel to 1 cement. The top finish was not used, but the concrete was straight-edged to proper level and troweled as smooth as possible. (See Fig. 56.)

Account 9003.52—Roof—Tar.

This account covers the cost of the material below, and the labor used in applying it to the roof. A composition of tar, cement and coal oil was made and painted directly upon the concrete roof in an effort to make it waterproof.

11 barrels of tar	$113.74
7 sacks cement	4.95
54 gallons coal oil	6.48
Miscellaneous	2.56
	$127.73

Account 9003.53—Roof—Down Spouts and Tile Drain.

This covers the cost of the material used in 10 down spouts and drains, together with the labor employed in erecting them. The down spouts are 4-in. galvanized iron spouts which, after leaving the gutters, pass directly through a hole prepared in the building wall to the inside of the building, and thence to the basement floor. At this point they enter a 4-in. vitrified sewer pipe which is laid beneath the floor, discharging into the drain down the center of the building. The material account stands as follows:

355 ft. 4-in. vitrified sewer pipe	$117.90
550 ft. 4-in. galvanized iron drain pipe	90.87
Miscellaneous	31.67
	$240.44

Account 9003.54—Roof—Painting Underside.

This account covers the cost of the paint material, brushes, scaffolds, etc., together with the labor required to paint the underside of the

power-house roof. High swinging scaffolds from the roof truss purlins were used to work from. They were slow and difficult to move from place to place. The rust was cleaned off and it was given two coats of white lead and linseed oil, cream color.

Account 9003.55—Roof—P and B Roofing.

This account covers cost of material and labor incident to applying a paper roof. Wood strips were imbedded in the concrete around the base of each ventilator, and across the roof at the juncture of the lean-to roof with the main building roof. A cement coating was then applied to the concrete followed by application of hot maltha, with sheets of felt imbedded in the maltha. The felt was so lapped one piece upon another as to give three thicknesses over the entire roof. Another application of hot maltha was swabbed over this ground work, followed by one course of three-ply P and B roofing paper.

Account 9003.60—Painting Sash.

This account covers the material and labor incident to painting all the power-house sash three coats. Linseed oil and white lead was used.

Account 9003.61—Painting Woodwork.

This account covers the painting of the power-house doors and miscellaneous woodwork.

Account 9004—Crane.

This crane has a capacity of 20 tons. It is operated by hand and spans the power-house floor, a distance of 50 ft. It has a 40-ft. lift and runs on 50-lb. rails. The account covers the material as shown below, and the labor of erecting the crane:

One 20-ton hand traveling crane	$1,278.00
Freight on same	408.00
Miscellaneous	37.27
	$1,723.27

Account 9005—Well Grading.

This covers the cost of grading off a point of conglomerate rock in preparing a site for a well. Large blasts of dynamite were used.

Account 9005.1—Shaft Sinking.

This covers the cost of sinking a shaft in conglomerate at the water's edge upon the site prepared by account 9005. The shaft was 6 ft. by 8 ft. and went to a depth of 45 ft. It was necessary to install and run a No. 7 Cameron pump to handle the water.

Account 9005.2—Timbering.

The timbers used were 8 in. by 8 in., Oregon pine square-shaft sets, making two compartments in the shaft. The shaft was lagged.

Account 9005.31—Aldrich Pump Installation.

The money expended under this account was for unloading two pumps which were not installed.

Account 9006.01—Nordberg Blowers—Foundation.

These two concrete foundations were each about 30 ft. by 15 ft. by 20 ft. The concrete was mixed 6 parts sand and gravel to 1 cement by machine, and transported 50 ft. to place. 100 per cent. of the vertical surface was formed. Each foundation had over thirty bolts set exact with template and piped. The pipes were in short pieces pulled up as the foundation raised and out at the completion. The cost of these pipes and the bolts are in every case given in the concrete cost. (See Fig. 49.)

Account 9006.1—Nordberg Blowers—Cost and Installation.

This account covers the cost of the material as listed below, together with the labor of erecting the same. These engines are two Nordberg cross-compound blowing engines, designed to compress 10,000 cu. ft. of free air at an altitude of 3,500 ft. to 12 lb. pressure, while 15 lb. may be carried if desired. The high-pressure steam cylinder is 20 in., the low-pressure 42 in., while the air cylinders are 44 in., all having the common stroke of 42 in. The engines are furnished 160 lb. steam pressure, superheated 75° F. The speed is 71 r.p.m. The labor of grouting, and the labor of testing out and starting up are included here.

2 Nordberg blowing engines, with receivers	$30,967.34
2 No. 34 Crane tilt traps	107.78
Grout, etc	1,438.90
	$32,514.02

Account 9006.2—Nordberg Blowers—Painting.

This account covers the cost of material and labor of painting the two Nordberg blowing engines. All of the power-house machinery was painted by contract for the sum of $820. This sum covered the labor and all tools, such as brushes, putty knives, light ladders, etc. The paint, oils, colors, driers, and scaffolds where necessary were furnished by the company. The money covered by the contract and material used was apportioned to the painting account of the different pieces of machinery on the basis of the time spent on each piece of machinery. Every machine was given one coat of paint, one coat of filler, and two coats of olive-green enamel.

Account 9007.01—Turbines—Foundation.

This concrete is identical with 9006.01.

Account 9007.1 Turbines—Cost and Installation.

This account covers the purchase price of three Curtis turbines and material as listed below, together with the labor of erection, grouting, wiring from generator to switchboard, testing and starting up. The turbines are 2,000-kw. Curtis-type horizontal shaft engines and direct connected to 2,500-k.v.a., 6,600-volt, 60-cycle, 3-phase, 1,900-r.p.m. generators. The approximate size of each unit is 23 ft. 8 in. long by 10 ft. 6 in. wide by 9 ft. 7 in. high, with a net weight of 108,300 lb.

3 turbines	$77,828.10
486 gallons of gargoyle turbine oil	233.04
Grout, electrical material	1,525.35
	$79,586.49

Account 9007.2—Turbines—Painting.

(See account 9006.2.)

Account 9007.3—Turbines—Air Pipe Making.

(See Fig. 57.) This account covers the making of the air ducts for the three turbines. They were fabricated in the smelter shops of No. 16 steel with 2½ by 2½ by ¼ angles. The total length for the three was 103 feet.

Account 9007.4—Turbines—Air Pipe Erection.

This account covers the labor of erecting the air ducts in account 9007.3. The material used was cloth insertion packing, rivets, hangers, anchors, etc.

Account 9007.51—Transformer Trucks and Transfer Table.

This account covers the placing of 325 ft. of 40-lb. rail for installing 15 transformers with trucks, and the cost of those trucks. (See Fig. 58.)

Account 9007.52—Auto Transformers.

This account covers the cost of the 10 transformers, oil and wire as listed below, together with placing wiring, trying out and testing the same.

10 oil-cooled auto transformers for raising voltage from 6,600 to 13,200 volts, G.E. Type "H," 60 cycles, 417 k.v.a. Y connected (3 transformers to a turbine and one spare)	$11,801.81
Oil and wire	243.10
	$12,044.91

Account 9008.01—Condensers—Foundation.

These were plain foundations machine mixed, in the proportions of 1 cement to 6 sand and gravel. The vertical surfaces were 100 per cent. formed.

Account 9008.1—Condensers—Cost and Installation.

This covers the cost of 3 Alberger surface condensers and the labor of placing and grouting them in position. Each condenser has 7,600 sq. ft. of surface.

3 Condensers	$19,436.04
Grouting, etc	127.51
	$19,563.55

Account 9008.2—Condensers—Painting.

(See account 9006.2)

Account 9009—Jet Condenser—Hot Well Excavation.

This covers the cost of making small excavations for a few piers in red clay with pick and shovels.

Account 9009.01—Jet Condenser—Hot Well Foundation.

This covers a small amount of concrete for piers, hand mixed—6 sand and gravel to 1 cement.

Account 9009.02—Jet Condenser—Hot Well Supporting Structure and Tank.

This account covers the cost and erection of 5.76 tons of steel. There was a quadrangular tower 19 ft. 6 in. high, with about 12 ft. base, surmounted with a 10 ft. diameter by 8 ft. 6 in. high steel tank. It was furnished by the Kansas City Structural Steel Co.

Account 9009.03—Jet Condenser—Cost and Erection.

This covers the cost of one 28-in. Alberger type "F," barometic jet condenser and erection above the tank of account 9009.02.

Account 9009.12—Jet Condenser—Dry Vacuum Pumps.

These air pumps remove the air from the barometic condenser and are located in the power house. The account covers the cost of the material listed below and the labor of erecting the same.

Two 15-h.p. slip ring motors, 440 volts, 3 phase, 60 cycles, 565-r. p. m., with resistance controllers	$739.92
Two 16 by 12 single-stage Alberger dry vacuum pumps	1,888.22
2 circuit breakers	39.88
Grout, cable, condulets, etc	191.99
	$2,860.01

Account 9009.13—Jet Condenser Dry Vacuum Pump—Painting.
(See account 9006.2.)

Account 9009.21—Circulating Pumps—Foundation.
These are about 15 ft. by 20 ft. by 10 ft. and are similar to 9006.01 in other respects.

Account 9009.22—Circulating Pump—Cost and Erection.
These pumps furnish the circulating water for the barometic condenser. The cost here includes the price of the material listed below and the labor of installing the same.

Two 35-h.p., 440-volt, 60-cycle, 570-r.p.m. motors,.....	$1,087.50
Two 2 lobe cycloidal pumps, 14 by 12, 17.8 gal. per rev.....	2,341.41
2 oil switches, 660 volt.....	39.89
Miscellaneous.....	66.88
	$3,535.68

Account 9009.23—Circulating Pumps—Painting.
(See account 9006.2.)

Account 9010.01—Air Compressor—Foundation.
This concrete is 10 ft. by 20 ft. by 15 ft. high. In other respects it is similar to 9006.01.

Account 9010.02—Air Compressor—Erection.
This account covers only the erection at the smelter of the following Ingersoll-Rand two-stage compressor. It was brought from the mines and erected at the smelter power house. The compressor has a steam-driven cross-compound Corliss engine. The steam cylinders are 13 in. and 22 in. and the air cylinders are 22 in. and 13 in. and the common stroke is 36 in.

Account 9010.04—Air Compressor—All Piping Except Steam.
This account covers the cost and erection of all the piping to the Ingersoll-Rand compressor except the steam piping.

Account 9010.05—Air Compressor—Wrecking and Transportation.
This account covers the labor of tearing down this compressor at Morenci, loading it on cars and the freight to the smelter. Such material is charged as was incident to these operations.

Account 9010.06—Air Compressor—Installation of Air Receivers.
This account covers the labor of installing a small air receiver in the power-house basement, for the compressor. No charge was made for the receiver.

Account 9011.01—Two Exciters, Two Air Pumps, Two Circulating Pumps—Foundation.

This is a large foundation about 15 ft. by 20 ft. by 10 ft. In other respects it is the same as account 9006.01.

Account 9011.02—Two Exciters—Cost and Installation.

This account covers the cost of the material listed below and the labor of installing the same. These are the exciters for the turbine generators. They are Ridgway tandem, compound, balanced, slide-valve engines, direct connected to 75-kw., 125-volt direct-current generators with a speed of 275 r.p.m.

2 exciters	$5,744.96
Cable wire, etc	286.28
Miscellaneous	87.02
	$6,118.26

In the labor cost is included wiring and connecting the machines to the switchboard, as well as erecting, grouting and trying out.

Account 9011.03—Three Dry Vacuum Pumps—Cost and Installation.

These pumps are for the surface condensers. The account covers their cost, erection, grouting and trying out. The three weighed 14,000 lb.

3 dry vacuum pumps 8-in. steam by 20-in. air by 12-in. stroke.	$3,136.11
Grout, packing, etc	53.99
	$3,190.10

Account 9011.04—Three Circulating Pumps and Engines—Cost and Installation.

These pumps furnish the circulating water for the surface condensers. This account covers the cost of the material listed below and the labor of erecting and trying out.

Three 2-lobe, 18 by 20, cycloidal pumps, capacity 49.5 gallons per rev., and three 27-in. flexible couplings	$4,425.25
Three 11 by 14 Ridgway, simple, balanced, slide-valve engines for direct connection to above pumps	4,124.69
Grout, packing, etc	179.43
	$8,729.37

Account 9011.05—Two Exciters—Painting.

(See account 9006.2.)

Account 9011.06—Three Air Pumps—Painting.

(See account 9006.2.)

Account 9011.07—Three Circulating Pumps—Painting.

(See account 9006.2.)

Account 9012.01—Two Motor Generators, One Air Pump, One Circulating Pump—Foundation.

These foundations are 28 ft. by 18 ft. by 11 ft. In other respects they are similar to 9006.01.

Account 9012.02—Two Motor Generators—Cost and Installation.

This account covers the material listed below as well as the labor of unloading, erecting, grouting, wiring to switchboard, and trying out.

Two 150-kw. synchronous motor-generator sets to supply 250 volt d.c.	$6,450.16
Conduit and wire	317.36
Miscellaneous	62.81
	$6,830.33

Account 9012.05—Two Motor Generators—Painting.

(See account 9006.2.)

Account 9013—Transfer Table Pit—Concrete.

This is principally a plain concrete slab, 8 in. thick, mixed by hand in the proportions of 6 sand and gravel to 1 cement. About 10 per cent. of the vertical surface was formed. It was chuted to the basement and wheeled in barrows 100 ft. to place.

Account 9013.01—Switchboard—Concrete Compartments.

This is a concrete switchboard, the large dimensions of which are 40 ft. 4 in. long, 4 ft. wide and 13 ft. 2 in. high. Down the center of the board is a 4-in. wall and on each side are 30 pockets made with 2-in. dividing walls. Reinforcing was done with Clinton wire mesh in the 4-in. wall and $\frac{1}{4}$-in. and $\frac{3}{8}$-in. rods were used elsewhere. A great many bolts and insulators were set in the board. The entire board was cast sectionally in place, using 5 parts of sand and gravel to 1 part cement. This cost includes rubbing down the concrete with pumice stone and filling all the air bubble holes and small voids.

Account 9013.02—Switchboard—Cost and Erection.

This account covers the material price of the secondary switchboard slabs. The primary or concrete switchboard construction, however, is in account 9013.01. Here, too, is the material price of the entire equipment for all of the switchboards, both primary and secondary, and the labor of installing the same.

Account 9014—Steam Piping North and South Mains—Excavation.

This covers the excavation for numerous piers done with pick and shovel and cast to one side.

Account 9014.01—Steam Piping North and South Mains—Foundation.
This concrete composes the piers which support the long structural steel steam pipe supports. They were part mixed by machine and part by hand, in the proportions of 6 sand and gravel to 1 cement. About 50 per cent. of the vertical surface was formed.

Account 9014.02—Steam Piping North and South Mains—Steel Supporting Structure.
In these steam-pipe trestle supports 11.8 tons of corrugated iron and 75.01 tons of structural steel were used. (See Fig. 60.)

Account 9014.03—Steam Piping Mains—Hangers and Anchors.
This covers the cost of material and making of all hangers and anchors used for the steam piping between the boilers and the machines in the power house. The hangers were made of ¾-in. rods and ½-in. by 2½-in. clamps running around pipe. Anchors were of same material fabricated to suit conditions surrounding place used. (See Figs. 60 and 61.)

Account 9014.04—Steam Piping—Cost and Erection.
Under this account all the material listed below is costed, together with the labor of its erection. These pipes run from the boilers to the power house in duplicate, making a complete loop about 1,120 ft. around. The main lines are 10 in., branches from boilers 8 in. and branches to engines of suitable size ranging from 4 in. to 8 in. All the joints are Van Stone, all valves and fittings are of cast steel. The line is required to stand 180 lb. pressure with 100° F. superheat. The gaskets used are corrugated bronze. The 10-in. lines are fitted with six 10-in. Harter expansion joints. (See Figs. 60 and 61.)

Six 10-in. Harter expansion joints	$1,684.77
One 6-in. cast iron separator	126.55
Two 10-in. cast steel vertical separators	843.47
One 10-in. cast steel horizontal separator	372.48
Two 6-in. separators and receivers (bot. outlet)	591.77
One 5-in. cast steel separator and receiver (bot. outlet)	261.40
Three 4-in. cast steel separators and receivers (bot. outlet)	687.43
Two 4-in. cast steel separators and receivers (bot. outlet)	476.28
Corrugated bronze gaskets	251.93
Ten 8-in. Lagonda valves	1,315.52
Twelve 10-in. gate valves	2,079.00
Two 34-in. and one 33-in. Crane tilt traps	143.69
Best Mfg. Co. pipe and fittings	8,738.89
Extra pipe and fittings	526.18
Miscellaneous	522.89
	$18,622.25

Account 9014.05—Steam Piping—Covering and Erection.

Part of this work was contract by the people who furnished the material and part was force account. Therefore the material figure given covers a large portion of labor. The total unit cost is the only valuable unit. The steam lines and all fittings were all covered with 85 per cent. magnesia blocks of double standard thickness, wrapped with 6-oz. duck. All of the line was then painted two coats—cream color.

Account 9015—Exhaust Pipe—Cost and Erection.

This covers the cost of the material as listed below and the labor incident to installing it. Some of the piping is cast iron, designed for a vacuum of 14 lb. per square inch. The rest of the piping is lap-welded wrought steel with cast-iron fittings. The installation covers the three 20-in. atmospheric exhausts from the turbines, as well as the exhaust from the blowers, compressors, exciters, engines, and circulating pump engines, to the jet condenser. It covers likewise the connections between the exhaust of the dry vacuum pumps, exciters, engines, surface condenser circulating pumps and the heater house. The pipe ranged in size from 3 in. to 42 in.

Three 42 by 13 Wainwright turbine expansion joints....	$656.70
Three 20-in. atmospheric relief valves.....	804.50
Three 42-in. low-pressure flanged base elbows.....	1,428.61
Three special 8-in. emergency stop valves.....	234.36
One 14-in. automatic atmospheric exhaust relief valve...	123.27
Pipe and fittings.....	4,585.74
Miscellaneous.....	882.48
	$8,715.66

Account 9015.01—Exhaust Pipe—Painting.

All exhaust pipe was given one coat of green silica graphite paint.

Account 9015.05—Exhaust Pipe—Covering and Erection.

The exhaust pipe from the engines in the power house to the heater house were all covered with 85 per cent. magnesia single standard thickness, wrapped in 6-oz. duck. Where the magnesia is exposed to the weather, it is wrapped with No. 28 galvanized iron. This account covers the labor and material incident to the above work. (See Fig. 62.)

Account 9015.10—Air Piping—Cost and Erection.

This is not a valuable cost.

Account 9015.11—Air Pipe—Painting.

This covers the painting of 9015.01. It is of no value.

Account 9015.2—Exhaust Pipe—Foundation.

This covers a number of small concrete piers.

Account 9015.21—Exhaust Pipe—Supporting Structure.

This account covers the cost of a number of exhaust pipe hangers, supports and staging used in erecting exhaust pipe. It is not a good cost.

Account 9015.22—Exhaust Pipe—Excavation.

Account 9016—Water Pipe—Excavation and Backfill.

This account covers the excavation of a trench about 3 ft. deep, through red clay and boulders, for a 16-in. wood stave pipe and the backfill after the laying of the pipe.

Account 9016.01—Water Pipe—Cost and Erection.

This account covers the cost of all the material listed below and the labor of its installation. Here is listed all the water piping about the power house; the 30-in. cast-iron suction pipe line from the cooling tower to the pumps; the 20-in. wooden lines from the pumps to the equalizing tank; the 16-in. wooden line from the jet condenser to the cooling tower, and the 16-in. wooden lines from the equalizing tank to the cooling tower; the 12-in. cast-iron lines from the circulating pumps to the jet condenser; the 8-in. line from condenser to condensed water pump house; the 6-in. line from condensed pump house to heater house, etc. (See Fig. 63.)

1,998.7 ft. 4-in. machine banded redwood pipe with collars. (Not used at New Smelter)	$397.74
354.6 ft. 20-in. machine banded redwood pipe with collars	365.24
1,104.2 ft. 16-in. machine banded redwood pipe with collars	861.28
22 flanged couplings	590.00
Freight on the above items	632.00
Two 12-in. check valves	97.00
Four 12-in. gate valves	172.00
Three 20-in. gate valves	283.50
Freight on the above items	176.38
Three 20-in. flanged, iron body, bronze mounted, double gate valves	403.49
Five No. 20 gauge copper plates	36.28
Two cast-iron bell and flange fittings, 6 bell bends	81.11
Freight and patterns on above	78.00
220 lb. cloth insertion packing	91.50
Best Mfg. Co. pipe	9,668.92
Pipe, fittings, misc'l material	2,503.44
	$16,437.88

Account 9016.02—Water Pipe—Painting.

This covers the painting of the pipe in 9016.01 that was above the ground.

Feed Water Heating Plant

Account 9017—Excavation.

This covers the excavation of the sump and piers, of the feed water heating plant. The material was red clay, with boulders and sand and gravel. The work was done with pick and shovel, cast into carts and hauled 300 ft.

Account 9017.01—Foundation.

This account covers some miscellaneous concrete cast as piers, low reinforced walls, 4 in. to 8 in. thick, and floor slabs. The walls and slab were reinforced with ⅜-in. rods on 6-in. centers. The concrete was hand mixed in the proportion of 6 sand and gravel to 1 cement and wheeled 50 ft. in barrow to the forms. Seventy-five per cent. of the vertical surface was formed.

Account 9017.015—Reinforced Floors and Stair Treads.

These floors were cast over I-beams in the building structure by using forms which allowed the encasing of the I-beams with concrete. The stairs were structural steel with channel treads. The treads were filled with concrete. The floor and the treads were reinforced with ¼-, ⅜- and ½-in. rods variously used and spaced. The concrete was hand mixed in the proportions of 5 sand and gravel to 1 cement, hoisted from the ground 20 ft. to place by means of a small air hoist. (See Fig. 65.)

Account 9017.016—Waterproofing Concrete Tanks.

This consisted of plastering the inside of the concrete tanks at the heater house with the following mixture. One cubic foot cement, 2.5 cu. ft. sand, 1 lb. Medusa waterproof compound mixed with sufficient water. The account covers the labor and material.

Account 9017.02—Steel Structure.

There were 26.63 tons of structural steel in this building.

Account 9017.021—Distribution and Equalizing Tank.

This account covers the material, labor of fabrication and erection of this tank. Its dimensions are 13 ft. by 13 ft. by 5 ft. high. It has 8 holes in the bottom averaging 20 in. in diameter. It is located on a concrete floor base 23 ft. off the ground. The weight of the tank is 4,800 lb. A material list shows the following. (See Fig. 69.)

17 sheets steel ¼ in. by 48 in. by 120 in	$192.99
4 pieces angle iron 3 in. by 3 in. by ⅜ in. by 20 ft. long	15.15
1,110 b.m. lumber	31.55
Rivets, bolts, nuts, etc	20.89
	$260.58

Account 9017.03—Tile Work.

The walls were started on the concrete floor of the feed water heating plant at an elevation of 26 ft. from the ground. The material was hoisted up by means of a small air hoist. In other respects this account is similar to the tile work elsewhere about the plant.

Account 9017.031—Unloading Tile.

Same as 9003.11.

Account 9017.032—Coping.

Same as account 8703.22.

Account 9017.033—Sills and Lintels.

The concrete sills and lintels for this building were separately molded as 8821.3, which see. The sills are 3 in. by 8 in. by 3 ft. 9 in. and the lintels 4 in. by 8 in. by 3 ft.

Account 9017.034—Painting Tile Walls.

The tile walls were given one coat of Bay State cement paint after the small holes had been filled and mortar cleaned from the face of the brick.

Account 9017.035—Doors, Windows and Frames.

This account covers the cost of the doors, windows and frames set in place. A segregated material list is as follows. (See Fig. 70.)

3 doors 2 ft. 10 in. by 6 ft. 10 in. by 1¾ in. with frames	$20.86
7 windows, 2 ft. 9¾ in. by 5 ft. 9 in. by 1¾ in. with frames	59.88
12 sets Caldwell sash balances, nails, locks, etc	19.24
	$99.98

Account 9017.04—Roofing.

This account covers the material listed below, and the labor of installing the same. The wood sheathing was nailed directly to nailing strips attached to the purlins. The paper was laid over the sheathing. (See Fig. 72.)

2,166 b.m. 2 by 8 Oregon pine S1S1E	$47.60
10 squares asbestos roofing	49.00
Miscellaneous	13.28
	$109.88

Account 9017.045—Ventilators.

This account covers the material used to make two 3 ft. 6 in. diameter ventilators, and the labor of installing one. The ventilators were made of No. 18 gauge galvanized iron. (See Fig. 71.)

Account 9017.05—Treating Tank—Concrete.

This is a concrete tank 9 ft. in diameter by 31 ft. 3 in. high, with a conical bottom, the tank is set directly upon the ground. The concrete

was mixed by hand 5 parts sand and gravel to 1 cement, and hoisted to place. The wall of the tank is 5 in. thick suitably reinforced in both directions with $\frac{5}{8}$-in. rods. Two sets of forms were used each 3 ft. 3 in. high. This cost covers all the material and labor of building this tank save the waterproofing in 9017.016. (See Fig. 68.)

Account 9017.06—Receiving Tank No. 1.

This is a concrete tank 6 ft. 6 in. high with 5-in. walls, suitably reinforced in both directions. The tank rests on a concrete floor about 23 ft. from the ground. The concrete details are the same practically as 9017.05. (See Fig. 67.)

Account 9017.07—Receiving Tank No. 2.

This concrete tank is 12 ft. in diameter, 6 ft. high, with 5-in. reinforced wall, supported on reinforced concrete columns 7 ft. high. The columns are here included. In other respects the tank account is similar to 9017.06. (See Fig. 66.)

Account 9017.08—Calibrating Tank.

This account covers the material, fabrication, and cost of erecting the following tank:

1 tank 54 in. high, 6 ft. diameter, of $\frac{3}{16}$-in. plate.

Account 9017.081—Tipping Meter.

This account covers the cost of material as listed below and the fabrication and erection of the same. The tipping meter is set directly over No. 2 receiving tank, in a wood frame attached to the top of the tank. The meter is shown in sketch No. 136.

Lumber, 2 by 4's, 6 by 8's, etc., 876 b.m	$24.47
Galvanized iron No. 10 and No. 12	112.03
2 by 2 by $\frac{1}{4}$ angle iron	3.69
One 6 wheel revolution counter	34.74
Bolts, washers, nails, etc	52.57
	$227.50

Account 9017.09—Heater and Recorder.

This account covers the material listed below and the labor of installing it on the concrete floor of the feed water heating plant, 26 ft. off the ground. The magnesia blocks are the covering for the heater.

	Factory	Freight	Clifton
1 No. 760 Cochrane feed water heater and receiver	$776.75	$204.48	$981.23
1 Wainwright closed feed water heater	775.00	74.64	849.64
One 6-in. Lea recorder and extra float valve	589.00	123.51	712.51
One 6-in. float for receiving tank No. 2			44.60
600 sq. ft. magnesia blocking 1½ in. thick			161.03
300 lb. magnesia cement			22.53
Miscellaneous			42.30
			$2,813.84

Account 9017.1—Sewer Excavation and Backfill.

This was a long trench about 3 ft. deep through red clay and boulders. Both excavation and backfill are here taken care of.

Account 9017.11—Sewer Pipe—Cost and Laying.

This covers the cost of 100 ft. of 24-in. vitrified sewer pipe, cement and miscellaneous material, together with the labor of laying the same.

Account 9017.12—Lighting.

This covers the cost of wiring for lights in the feed water heating plant.

Account 9017.13—Painting.

This covers the cost of painting the underside of the roof, doors, all frames, and window sash. It was done with two coats of white lead and linseed oil, cream color.

Account 9017.14—Wood Walkway and Tank Covers.

This covers the cost of material and its installation as listed below:

Lumber 1,564 ft. b.m.	tank cover 1 by 6 O.P. S1S1E rails 2 by 3 and 3 by 3 S4S sills, 2 by 8 ROP 20 in. centers on tank walkways 2 by 8 O.P. S1S1E

Account 9017.115—Alterations.

Condensed Water Pump House

Account 9017.20—Excavation.

This was a side hill cut. It includes a backfill made later inside the building foundation for a concrete floor base. The excavation was in red clay, with boulders and sand and gravel. It was done with pick, shovels and wheelbarrows.

Account 9017.21—Foundation.

This concrete was cast as the walls of some square tanks. It was reinforced, mixed by hand in the proportions of 5 sand and gravel to 1 cement. One hundred per cent. of the vertical surface was formed. (See Figs. 73 and 74.)

Account 9017.2—Floor.

This concrete covers the small building walls, large side hill wall, floor slab and tank bottom. Three-eighth inch and three-fourth inch rods were used as reinforcing. One hundred per cent. of the vertical surface was formed. Hand-mixed concrete, 5 sand and gravel to 1 cement, was used.

Account 9017.24—Doors, Windows and Frames.

This account covers the cost of material below with its installation.

2 windows 2 ft. 9½ in. by 1½ in. glazed, with factory ribbed glass.	$6.51
2 frames for above	9.28
1 door 2 ft. 10 in. by 6 ft. 10 in. by 1¾ in.	3.02
1 door frame	3.87
	$22.68

Account 9017.242—Tile Work.

This covers the cost of tile, mortar, mason and carpenter labor and hauling. The tile was handled 400 ft.

Account 9017.243—Coping.

Same as 8703.22.

Account 9017.25—Roof.

This account covers the material entering into the roof frame, as well as the sheathing paper and labor of installing the same. The rafters were 2 by 10, with 2 by 8 sheathing tongued and grooved, covered with asbestos paper. (See Fig. 75.)

984 ft. b.m. lumber	$33.49
5 rolls asbestos composition roofing paper	24.50
Nails, etc	2.84
	$60.83

Account 9017.26—Pumps and Piping.

This account covers the purchase price of the material listed below and the entire labor cost required in installing the same.

2 Goulds No. 2½ single stage, single side suction, centrifugal pumps, arranged for direct connection, capacity of pump 200 gallons per minute, against 80 ft. head Two 10-h.p. induction motors, 1,710 r.p.m., with starters	$530.83
Grouting, pipes, fittings, valves, etc	160.54
	$691.37

Account 9017.27—Lighting.

This covers the lighting of the condensed water pump house.

Power House Miscellaneous Accounts

Account 9018.1—Power and Lighting Transformers.

This account covers the material as listed below, with the labor charge of installing the same, the labor of unloading, erecting, connecting up, drying out, and testing the transformers noted. These are located in the basement of the power house.

2 Burke air break switches	$134.97
Four 200-k.v.a. transformers, 13,200-volt, 440-volt	2,999.80
Two 20-k.v.a. transformers, 440-volt, 220-volt	304.40
One 20-k.v.a. transformer, 440-volt, 110-volt	152.18
Freight on transformers	90.48
Wire, conduit, electrical material	614.04
Transformer oil	282.34
Miscellaneous	136.12
	$4,714.33

Account 9019—Lighting.

This account covers the material and labor of installing the lighting in the power house. The work was all in conduit.

Forty-eight 250-watt tungstens with reflector.
Forty-six 16-c.p. carbon lamps.
610 ft. 1½-in. conduit with weatherproof wire No. 12.
1,710 ft. 1-in. conduit with No. 12 weatherproof wire.
25 ft. brewery cord.

Account 9020—Power House Oiling System.

This covers the cost and labor of installing the power house oiling system. Two 60-gallon feed tanks are so located that the oil is piped to all the engines by gravity. The waste flows to a water separator, thence through filters into a closed tank, from whence by air pressure it is forced up to the feed tanks.

Account 9021—Benches, Bolt Racks, Etc.

Under this account were built a number of benches, a rack for bolts, a telephone booth, wrench board, and a number of miscellaneous carpenter jobs about the power house including the replacing of about 50 broken panes of glass. The material account stands as follows:

Ceiling lumber	$8.23
Common lumber	31.24
Glass	6.39
Miscellaneous	5.94
	$51.80

Account 9022—Instruments and Gauges.

This account covers the purchase price of the following material, and the labor of installing the same:

3 Tagliabue vacuum gauges	$92.58
10 Tagliabue thermometers	92.85
1 vacuum gauge	30.86
1 Bristol pressure gauge	25.30
1 Bristol gauge	22.95
1 Bristol gauge	21.35
1 clock	46.63
Miscellaneous	10.54
	$343.06

Cooling Tower

Account 9050—Excavation.

This excavation entailed the making of a surface cut and two long trenches. The material encountered was red clay filled with caliche. It was done with pick and shovel and handled in carts and wheelbarrows.

Account 9050.01—Backfill.

This backfill was made to bring the ground up to proper level for the cooling tower floor. Fresnos and scrapers were used to transport the dirt which was tamped in 4-in. layers.

Account 9051—Foundations—Sumps and Gutters.

This concrete was cast as a large number of piers about 1 ft. by 1 ft. by 4 ft. as a sump 35 ft. by 10 ft. by 13 ft. with reinforced walls 8 in. thick, and as two gutters 626 ft. long, in cross section about 3 ft. by 3 ft. having 4-in. reinforced concrete walls. The concrete was hand mixed; for the piers 7 sand and gravel to 1 cement; for the gutters and sump, 5 sand and gravel to 1 cement. In the sump $\frac{3}{4}$- and $\frac{5}{8}$-in. rods were used and in the gutters $\frac{1}{2}$-in. The vertical surfaces of all the above were 100 per cent. formed. (See Fig. 107.)

Account 9051.02—Floor.

Between the gutters of the cooling tower is a reinforced concrete slab about 28 ft. 6 in. by 600 ft. by 4 in. thick. Clinton wire cloth 86 in. wide with a 4 in. by 4 in. mesh was used. No top finish was used but the concrete was straight edged and troweled. It was mixed by hand, 5 sand and gravel to 1 cement, and wheeled 150 ft. to place, in barrows. (See Fig. 107.)

Account 9051.03—Waterproofing Concrete.

See account 9017.016.

Account 9052—Woodwork.

The tower is built of wood and is 626 ft. long, 35 ft. 6 in. wide at the base, by 20 ft. 6 in. high. Sketch No. 76 shows the design in other particulars.

Account 9053—Alterations.

Oil Supply Sump and Pump House

Account 9060—Excavation.

This excavation consisted of two deep cuts through red clay and boulders into sand and gravel. The work was done with plow, slips, picks and shovels.

Account 9060.01—Concrete.

This covers the making of a covered reinforced concrete sump, the general dimensions of which are 5 ft. by 10 ft. by 140 ft. with walls 8 in. thick—top and bottom slab 5½ in. thick, also a pump house about 20 ft. by 20 ft. by 20 ft. high, with walls 10 in. at top and 20 in. at bottom. The walls were reinforced with ¾-in. rods and the sump with ⅝-in. rods properly placed. The concrete was machine mixed in the proportions of 5 sand and gravel to 1 cement, wheeled in barrows an average of 125 ft. About 80 per cent. of the vertical surface was formed. (See Fig. 37.)

Account 9060.02—Pumps.

This account covers the cost of the following material and its erection in the pump house.

Two 5 by 8 Aldrich vertical triplex, single-acting pumps, 37 r.p.m. with metallic packing	$1,597.91
Two 10-h.p. induction motors, squirrel-cage, 3-phase, 60-cycle, 440-volt, 850-r.p.m	287.22
2 auto starters 2 overload releases calibrated from 6 to 18 amperes per terminal	124.36
Miscellaneous	26.09
	$2,085.58

Account 9060.03—Inlet Piping to Sump.

This covers the cost of the following material and its installation between unloading tracks and the oil sump. (See Fig. 38.)

Six 10-in. wrought pipes 18 ft. long	$85.54
Six 10-in. cast-iron ells	38.64
Miscellaneous	2.37
	$126.55

Account 9060.04—Lighting.

This covers the cost of material and the labor of installing four drop lights in the pump house.

Account 9060.05—Roof Steel Work.

This covers the cost of the following material and the labor of installing it. (See Fig. 39).

1,876 lb. 10-in. I-beams
320 lb. 8-in. I-beams

Account 9060.06—Doors, Windows and Frames.

This covers the cost of the following material and the labor of installing it in the pump house. (See Fig. 41.)

4 sash, 3 ft. by 2 ft. 6½ in. by 1¾ in	$17.79
2 sash, 3 ft. by 4 ft. by 1¾ in	14.05
2 sash, 2 ft. 6 in. by 4 ft. by 1¾ in	12.13
Lumber for doors and all frames	12.69
Hardware	8.36
	$65.02

Account 9060.07—Roof.

This covers the cost of the roof material as given below with the labor of placing the same. The "hyrib" was placed upon the steel of 9060.05, wired in place, covered with 2 in. of concrete, 5 sand and gravel to 1 cement, and plastered on top and bottom, with 1 part cement, 3 sand, ½ hydrated lime. Later it was covered with P and B roofing, see 9003.52. (See Fig. 39).

784 ft. No. 24 "hyrib"	$77.41
7 squares P and B roofing	48.39
Concrete plaster material	36.86
	$162.66

Account 9060.075—Ventilators.

This covers the cost of material in the making and placing of two ventilators shown in Fig. 40.

1 ventilator is 18 in. diameter, 40 ft. high made of No. 20 galvanized iron, using 130 ft. ½-in. guy wire.

1 ventilator is 18 in. diameter, 11 ft. high, made of No. 20 gauge galvanized iron

Two 500,000-Gallon Oil Tanks

Account 9060.10—Wrecking and Transportation.

This account covers the labor and material incident to taking down at Lordsburg, New Mexico, and transporting to Clifton, two 500,000-gal. oil tanks which had been in use there. The labor item is the cost of tearing down and loading these tanks. The material item is the freight on the tanks between Lordsburg and Clifton.

Account 9060.11—Excavation.

This account covers the making of a top slice to prepare the site for the foundation of the two 500,000-gal. oil tanks. It was done with plow, slips, pick and shovel and handled 150 ft.

Account 9060.12—Foundation.

This concrete was cast as two low circular walls, 1 ft. thick and 2 ft. deep, 62 ft. in diameter. It was mixed by machine, 8 parts sand and gravel to 1 cement, reinforced with two ¾-in. rods and hauled to place 150 ft.

Account 9060.13—Erection.

This covers the erection of the two 500,000-gal. tanks, part on force account by the Kansas City Structural Steel Co. and part by the Arizona Copper Co. It likewise covers the cost of giving them one coat of paint, testing them out, and calking. Here too are the material prices of the ventilators and gauges. The tanks were 60 ft. in diameter and 25 ft. high. (See Fig. 42.)

Account 9060.131—Roof Supports.

This account covers the cost of the material and labor of installing the same, used for supporting the sheathing over the two tanks. This material amounts to 10,000 ft. board measure for the two tanks. (See Figs. 42 and 43.)

Account 9060.132—Sheathing, Lath and Plaster.

Over the roof supports 1-in. sheathing was laid. On this No. 27 painted expanded metal lath was placed and plastered with a mixture of 1 cement to 3 sand, which was waterproofed with hydrated lime. Each tank has a cornice which ventilates by screened openings through its bottom. (See Figs. 42 and 43.)

Account 9060.14—Railroad Grading.

Along each side of the oil sump a large railroad grade was made. This account covers it.

Account 9060.15—Track Laying and Ballasting.

On the grade mentioned in 9060.14 the following track was laid. The account covers the material and labor of laying and ballasting.

802 white oak ties	$852.60
1 No. 9 frog	158.39
Miscellaneous track fastenings	82.00
	$1,092.99

Account 9060.16—Oil Track Bumpers.

This account covers the labor and material incident to making and putting in place two bumpers at the ends of the tracks on each side of the oil sump. The material is divided thus:

Lumber	$8.52
Iron bolts, nuts, etc	39.40
	$47.92

Account 9060.17—Bridges over Wood Pipe.

This covers four wooden bridges entirely buried in the fill over a number of wooden pipes. They were placed here to avoid the constant breaking of the pipes. The excavation and backfill are here included. (See Fig. 44.)

Lumber	$70.02
Bolts, etc	17.12
	$87.14

Oil Supply Tanks for Reverberatories and Boilers

Account 9060.20—Excavation.

This covers a deep cut for a retaining wall. It was in sand and gravel and made with pick and shovel. The material was hauled 300 ft.

Account 9060.21—Foundation.

This covers the concrete in a reinforced wall about 60 ft. long, 8 in. at top, 18 in. at bottom and 16 ft. high; ½-in. and ¾-in. rods were used for reinforcing. The concrete was machine mixed 5 sand and gravel to 1 cement and wheeled 75 ft. in barrows to place. One hundred per cent. of the vertical surface was formed.

Account 9060.22—Cost and Erection.

This covers the cost of eight 163-bbl. steel oil tanks, with roofs and ventilators, erected on their foundations.

Account 9060.23—Piping.

This covers the material and labor cost for 785 ft. of piping, varying in size from 1 in. to 6 in. diameter. Here are also included the fittings and valves. The piping connects the tanks with the pumps.

Account 9060.40—Piping Excavation.

This includes all the trenching and backfilling incident to the oil lines from the 500,000-gal. oil tanks to the small 163-bbl. tanks. The trenches were 2 ft. wide and about 3 ft. deep on the average.

Account 9060.41—Pipe and Laying.

This account covers the cost of the pipe enumerated below and the labor of laying it.

172 ft. 12-in. wrought-iron pipe
270 ft. 16-in. wrought-iron pipe
850 ft. 8-in. wrought-iron pipe
596 ft. 2½ in. wrought-iron pipe

1,888 ft. total

A 16-in. line runs from oil sump to pump house, also from pump house to storage tanks. The 8-in. line runs from pump house to the 163-bbl. tanks. The 2½-in. line runs from the Wilgus oil pumps to each of the reverberatories.

Account 9060.5—Heating Installation.

This account covers the material noted below and the labor required to install the same. This 2½-in. steam line is tapped off the steam line at the power house, run under ground through conduit and is packed in asbestos fiber. At the other end the pipe connects with a cast-iron oil heater.

1 cast-iron oil heater	$303.82
1 No. 33 Crane tilt trap	35.91
280 ft. 8-in. conduit	547.49
Asbestos	29.00
2½-in. pipe, fittings, etc	151.82
	$1,068.04

The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada

BY G. H. CLEVENGER, PALO ALTO, CAL.

(Salt Lake Meeting, August, 1914)

Continuing the discussion of the paper of James Johnston, presented at the New York meeting, February, 1914. See *Trans.*, xlviii, 3 to 32 (1914).

THIS paper cannot fail of being of great interest and value to all who are interested in the cyanide process, on account of the important developments which it records. Mr. Johnston is to be commended for the large amount of detail which he gives, and the company and its management for permitting the publication of data of this character, which while of great value to others could well be considered private property.

It might be discussed from a number of standpoints: First, as an example of recent practice in mill and cyanide plant construction; second, the mill practice; third, the cyanide practice; and fourth, the new features in cyanide treatment involved. Perhaps a more vivid way of expressing it would be: First, the equipment or tools with which the various operations are carried on, are they the best and most efficient which could have been selected for their respective uses; and, second, is the best use in every case made of these tools? It may be well inquired is the highest possible recovery of silver made at the lowest cost per ounce recovered? For after all, the making of the greatest ultimate profit is inseparably associated with successful metallurgical practice; there must therefore be at all times a careful balance maintained between recovery and cost of obtaining it.

I wish to call attention in a very general way to the equipment used and to an anticipation of questions which are bound to arise in this connection. However, the bulk of my discussion will be confined to the presentation of certain additional data, together with a more or less general discussion showing the line of development which leads up to the present practice and the applicability of certain features of it to similar problems elsewhere.

There are a number of vital points of broad general interest involved in Nipissing low-grade practice upon which metallurgists will by no means be agreed to which I wish to call particular attention in the hope that it will lead to the presentation by others of data of practice in other districts which will assist in reconciling this divergence of opinion.

It will be noted that the crushing equipment closely follows recent South African practice. The use of a silex lining in the tube mills might be questioned from the standpoint of economy; but quite aside from this point, upon which I am not prepared to give authoritative data, there are perhaps two distinct advantages possessed by this type of lining in this particular case. With the very fine grinding practiced at this plant there would be, with iron or steel liners, a much larger proportion of finely divided iron in the pulp. My experience with the treatment of the high-grade ore showed that the presence of iron interfered with the extraction of the silver. It is also a well-known fact that metallic iron in an extremely finely divided state may cause an important cyanide consumption. The installation of the old type of mechanical agitator will be questioned by many operators, particularly as there is an air lift used in each tank in addition to the mechanical stirring gear.

There has been considerable misconception regarding the character of the low-grade ore, and certain writers have made rather sweeping statements regarding the non-suitability of Cobalt ores for vacuum filtration, presumably upon the assumption that the low-grade ore had a heavy gangue similar to the high-grade ore. The specific gravity (2.70 to 2.72) of the low-grade ore is not very different from that of the siliceous ores of other districts. My own experience in making the large-scale tests was that it was a most favorable ore for vacuum filtration. This statement I do not wish misconstrued as meaning that the vacuum type of filter is unqualifiedly the best for Cobalt ores, for so far as I am aware comparative tests of the different types of filters have never been made in this district.

Briefly the status of metallurgical practice in the Cobalt district at the time that we began the investigation of the treatment of Nipissing ores was as follows: The high-grade ore was separated by sorting and jigging at surface plants, and sold to smelters outside the district, there not being up to the time of the advent of the Nipissing high-grade mill any of this ore treated locally. The low-grade ore, constituting the portion rejected in separating the high-grade ore, the dumps, and the low-grade material taken directly from the mine, was originally treated exclusively by concentration, the concentrates being sold to smelters. Later certain of the concentration mills provided cyanide annexes for the purpose of making a further saving from their tailings. Then came the building of mills which were primarily cyanide plants in which concentration was to be depended upon for the removal of the refractory minerals, and the major portion of the silver recovered as bullion through the medium of the cyanide process.

While the mills using the cyanide process unquestionably made a higher ultimate recovery, the mills which had confined their efforts to the development of their concentration practice, although realizing a lower recovery, appeared to be making as great and in some cases a greater ultimate profit than the mills employing a combination process. In other

words, the greater recovery, which necessarily required a greater capital expenditure, did not in all cases mean a greater ultimate profit.

Early in the course of our work we considered the treatment of the low-grade ore and in this connection made a number of small-scale tests. These clearly indicated that the ore could be treated by cyanidation, but they had not proceeded far enough for us to determine certain difficulties, and problems, which even at this early date we were quite sure would arise on account of the experience of others in the district. At this time we had not worked up to the maximum extraction or assured ourselves that concentration could be dispensed with. It might be mentioned that from the first we were determined to treat all the ore produced by the Nipissing company by hydrometallurgical processes and that nothing but refined silver was to leave the premises. Shortly after making the preliminary tests mentioned our efforts were centered upon developing and getting into successful commercial operation the process for treating the high-grade ore. This fully occupied our time until late in the summer of 1911, when the treatment of the high-grade ore was upon a well-established commercial basis, and we again turned our attention to the treatment of the low-grade ore. After a thorough study of the practice and results of other mills in the district, the management and ourselves were unanimous in the opinion that the issue lay squarely between the relative economy of straight concentration and a process, preferably as near straight cyanidation as possible, which would produce only refined silver. The middle ground involving concentration and cyanide treatment did not appear attractive for various reasons.

Generally the object of concentration in cases where it is practiced in conjunction with the cyanide process is to recover gold and silver not soluble in cyanide solution, or to remove interfering elements, but experience shows it to do this but imperfectly, and further, it invariably removes gold and silver which are readily soluble in cyanide solution, which could be to better advantage dissolved by the solution and recovered as bullion. In cases where it is feasible to concentrate after cyanide treatment this last objection of course is not valid; but this cannot be readily done, for it means repulping of the tailing and provision for a regular feed to the concentration devices; moreover, the conditions of fine grinding which make for the best results in cyanide treatment are not those most conducive to the best work in concentration. A more or less complete concentration plant means a greater capital expenditure and it further means the control of two separate operations which involve entirely different principles and methods of operation. Generally you are attempting to operate a concentrator and a cyanide plant with neither up to full efficiency. The conditions of the two processes are therefore at variance and it is the constantly growing opinion that concentration in connec-

tion with cyanide treatment should only be used where there is a distinct advantage to be derived.

Many small-scale tests were made involving various combinations of treatment. The results of these, together with the other data which had been collected in the district, led us to make the first large-scale tests along rather conservative lines, particularly as regards the removal of the native silver and dyscrasite (an antimonide of varying composition). The preliminary examination of these ores indicated that the silver occurred in three general forms: First, native silver, of greater or less purity, and dyscrasite, which on account of their coarseness would not readily pass into solution; second, silver minerals readily dissolved by cyanide solutions; and third, some combination of silver which resisted ordinary cyanide treatment and was dissolved only after extremely fine grinding and a long period of agitation.

In order to show the general character of the low-grade ore, the results of four large-scale tests made upon lots of ore representative of the type of ore produced by the Nipissing company, and involving a total of about 36 tons, are quoted from rather fully.

Each lot of ore was first sent to the sampler where it was weighed, crushed to 4-mesh and the assay sample separated. It was then transported to the point where the tests were to be made. It might be explained that these large-scale tests were run in the high-grade mill and the time available for them was determined by the time necessary to make certain necessary alterations and repairs to the mill. The tube mill had just been relined and therefore contained no amalgam. Agitator tanks, settler and other equipment were cleaned out very thoroughly, and before running the tests proper, low-grade ore was passed through the entire system. I have every reason to believe that in general there was no salting except in the case of perhaps four or five samples. At any rate, if there was any error the probability would be that the residues actually assayed higher than the true value, therefore the percentages of extraction noted would be lower than those to be expected in actual practice, when employing the same method.

The general method of procedure in each case was to grind the ore, 2 tons at a time, with a ton of solution, the proper amount of lime, and a part of the cyanide, for a period of 4 hrs. The time of grinding had been previously determined by grinding a sample of the ore in the tube mill for varying lengths of time and making sizing tests. The aim was to have practically everything pass a 100-mesh screen. This method of unit grinding had to be adopted on account of there being no arrangement for the ordinary method of continuous feed and discharge. After grinding, each charge was dumped into a settler containing 522 lb. of mercury, and after each charge the tube mill was rinsed out with cyanide solution which was also allowed to run into the settler. The oak-shod muller

was raised well above the bottom of the settler so that there would be no grinding effect upon the mercury. Each charge was agitated in the settler for a period of 1 hr. It will be noted that no mercury was used during the grinding. The idea was to give an opportunity for any particles of metallic silver or dyscrasite, which might have resisted grinding, to settle out so that they would not be carried down into the treatment tanks where they would cause trouble. The mercury at the bottom of the settler simply acted as a collector for these particles. Particles of native silver and dyscrasite fine enough to remain in suspension in the pulp during agitation were assumed to be fine enough for satisfactory extraction by solution in cyanide. My idea of this operation was that it was a mechanical method, involving settling rather than amalgamation, of separating a portion of the silver which either would not be dissolved in the treatment tanks or would require an undue amount of grinding to reduce it to a state of subdivision fine enough to be readily dissolved. After agitation in the settler the pulp was drawn off into agitator tanks of the ordinary mechanical type fitted with means of introducing air and so connected that the charges could also be agitated with a centrifugal pump. The progress of the treatment was recorded by determinations made upon samples taken every 6 hr. The effect of various forms of agitation was studied and in this connection pump agitation was found to be the most effective for the refractory silver minerals which dissolved very slowly at the last. A small amount of air was introduced during the whole agitation period. Pump agitation was not resorted to until at the last.

The following assays of the four lots of ore treated show the distribution of the silver:

	Lot A Ounces per Ton	Lot B Ounces per Ton	Lot C Ounces per Ton	Lot D Ounces per Ton
Pulp passing 100 mesh—(0.0055 in.).....	29.70	30.10	47.80	27.10
Metallics on 100 mesh.....	0.01	0.01	0.16	0.05
Metallics on 20 mesh—(0.0340 in.).....	....	....	0.19	0.07
Metallics on 8 mesh—(0.0900 in.).....	4.18	3.23	5.29	2.79
Metallics on 4 mesh—(0.1780 in.).....	1.24	1.33	0.46	0.98
Commercial assay.....	35.13	34.67	53.90	30.99
Correction on pulp.....	1.30	2.00	2.90	1.10
Corrected assay.....	36.43	36.67	56.80	32.09

The following analysis by Denny[1] of a sample of the low-grade ore will show its general composition.

[1] James Denny: Desulphurizing Silver Ores at Cobalt, *Mining and Scientific Press*, vol. cvii, No. 13, p, 487 (Sept. 27, 1913).

	Per cent.		Per cent.
Silver	0.106	Lead	0.064
Copper	0.270	Calcium oxide	9.020
Arsenic	1.880	Magnesium oxide	4.330
Iron	1.920	Aluminum oxide	10.030
Sulphur	0.640	Carbon dioxide	11.060
Bismuth	0.010	Mercury	trace
Nickel and cobalt	0.730	Insoluble	59.840

An analysis made by Johnson and Sons[2] upon a sample of the residues from the high-grade ore, shows antimony 3.8 per cent., bismuth 0.09 per cent., tellurium 1.39 per cent., and traces of tin, zinc, and magnanese. These elements are all undoubtedly present in the low-grade ore as there is, unavoidably mingled with it, more or less high-grade ore.

I have also observed the presence of a small amount of graphite in Nipissing ores.

The extraction of the silver by settling averaged 46.7 per cent., and the total extraction by settling and cyanide treatment at the end of 4 hr. of grinding and 1 hr. of settling averaged 66.5; 19.8 per cent. of the silver was dissolved during this period by cyanide. At the end of the first 6 hr. of agitation, the total extraction had reached 76.3 per cent. At the end of the next 30 hr. of agitation or a total agitation period of 36 hr., the extraction was 87.6 per cent. To raise the extraction 4.2 per cent. or to reach a total extraction of 91.8 per. cent. required a long additional agitation period. Of the 8.2 per cent. of silver remaining after long-continued agitation, 3.7 per cent. was soluble in hot dilute nitric acid. The average total percentage of silver which could be extracted by combined settling, cyanide treatment, and dilute nitric acid was 95.5 per cent. The proportion of silver extracted during 4 hr. of grinding and 1 hr. of settling with lots *C* and *D*, which were similar in character, was considerably higher than with lots *A* and *B*, but the extraction curves for *C* and *D*, as the cyanide treatment continues, straighten out and soon fall below the curves for *A* and *B*. A continued slow increase in extraction was particularly noted with lot *C*, which contains a somewhat higher proportion of silver. The peculiar character of these extraction curves is doubtless due, at least to a certain extent, to the presence of a large proportion of dyscrasite, the coarse portion of which settles readily, while the finely divided portion dissolves very slowly in cyanide solution.

Although these results clearly indicated that a cyanide plant would be most advantageous, it was the general consensus of opinion that more experimental work should be carried on to simplify and improve the proc-

[2] R. B. Watson: Nipissing High Grade Mill, Cobalt, *Engineering and Mining Journal*, vol. xciv, No. 23, pp. 1078, 1079 (Dec. 7, 1912).

ess, if possible. From the first, there was a strong prejudice against the use of mercury in the low-grade mill. This is best appreciated by those who have had experience in handling large amounts of mercury in mill practice. Although certain of my early small-scale experiments had not been very promising when it was attempted to treat the whole ore directly by cyanidation, Mr. Butters strongly adhered to the idea of grinding the whole ore, native silver, dyscrasite, etc., to such a degree of subdivision that all the silver could be dissolved by cyanide. Tests were accordingly made in this direction, and through the guidance of Mr. Butters, gradually led into the present system of treatment where the same idea of grinding the native silver and dyscrasite to a state of subdivision such that it can be dissolved by cyanide solutions is practiced, the desulphurizing treatment being only effective in altering the sulphides and sulpho-antimonides.

Now I wish to make this point very clear, for this problem of handling metallic gold and silver in ores which are cyanided is one that has often perplexed metallurgists and one that has resulted in a variety of solutions. The usual answer to this problem, although other methods have been used, is either amalgamation or concentration. On the one hand we have Mr. Butters, who has taken the extreme view that an ore containing native silver and dyscrasite, some of which will remain upon a 4-mesh screen, can be ground and cyanided direct; on the other hand we find some metallurgists amalgamating comparatively low-grade gold ores preliminary to cyanide treatment, where the actual weight of metallic gold to be ground would be many times smaller than the weight of the metallic silver in Nipissing ore and everything considered much easier to grind and dissolve than Nipissing silver.

Mr. Butters has demonstrated that his plan is possible with Nipissing ore, for the results being obtained in the mill bear testimony to that fact; but quite naturally, it will be asked at what cost? Is the ore ground excessively simply that the more resistant metallic silver and dyscrasite may be reduced fine enough to pass readily into solution and the whim of recovering everything by cyanidation be satisfied? It is true of course that selective grinding can be practiced to a considerable extent with heavy brittle minerals, but this is less true of the metallic silver and dyscrasite of Nipissing ore. By selective grinding, I mean grinding of an ore consisting of a relatively light gangue and a heavy *brittle* mineral so that the mineral becomes more finely divided than the gangue. This automatically takes place in a closed circuit of classifiers, other than screens, and a tube mill. This often causes a mineral to be ground down sufficiently fine for satisfactory extraction of gold and silver without excessive grinding of the gangue.

As a result of the first experiments which were made upon this ore, I called particular attention to the necessity of extremely fine grinding, and

this point has been confirmed by all who have since investigated its treatment. Now bearing in mind the necessity for this extremely fine grinding, presumably for certain of the brittle silver minerals, it is readily conceivable that when this necessity for fine grinding is satisfied in a closed circuit of tube mills and classifiers, all of the metallic silver and dyscrasite will also have been reduced to a degree of subdivision such that it readily passes into solution. After all, Mr. Butters may be right in this case, due to the formation of the ore by "Dame Nature" to fit his metallurgical ideas, or he may have recognized this peculiarity from the outset, and as all wise men should do, has taken advantage of it. Under other conditions than those which *possibly* obtain with Nipissing ore, such a procedure could readily lead to poor economy through the excessive grinding of worthless gangue in order that metallic gold or silver might be dissolved by cyanide solutions, and other methods of recovery dispensed with.

The so-called desulphurizing process is of interest as being the first large-scale application of the use of a reducing agent to effect certain chemical changes prior to cyanide treatment. To the thinking operator, it is indeed refreshing in this day of excessive oxidation and aeration in cyanide treatment, to note that at times there may be virtue in reduction. I might mention that this mania for oxidation has even gone so far that certain operators have aerated their pregnant solutions prior to precipitation. When it is remembered that precipitation is distinctly an operation of reduction, the fallacy of this course is apparent. That the deleterious effect of air upon precipitation is no mere theory, will be confirmed by operators using zinc-dust precipitation who have experienced air leaks in the suction of their precipitation pump.

The idea of using a preliminary treatment involving the use of aluminum and caustic soda in connection with the treatment of precious-metal ores is by no means new or original, as many have supposed. Over 10 years ago Walker[3] published an account of experiments made in this direction by himself and Martin. As cyanide operators generally are not familiar with this work, I will quote from it at some length.

"In an experiment designed to determine the effect of chlorine and cyanogen at the moment of liberation at the anode, ore was spread on a piece of platinum foil immersed in a solution of salt and potassium cyanide. The electrode intended to act as cathode was suspended from above. Through an error the feed wires were crossed and the platinum plate on which rested the ore was made the cathode. Almost immediately after turning on the current a purplish-red color was observed around the plate and subsequently a black cloud was thrown off. Upon removing the ore the gold was found securely plated upon the platinum, while the black cloud, when collected and analyzed, proved to be tellurium. To prove that the anions Cl and CN played no part in the

[3] Walker, W. H.: Note on the Electrometallurgy of Gold, *Transactions of the American Electrochemical Society*, vol. iv, pp. 51 to 53 (1903).

reaction caustic soda was substituted as electrolyte and identical results obtained. Evidently the gold was being recovered by cathodic reduction alone.

. . . *it was found that, in general, any method which subjected the telluride to the action of nascent hydrogen in an alkaline solution would effect the reduction; or to put it differently, whenever gold telluride in the presence of alkali came in contact with a metal from which hydrogen was being liberated, tellurium was set free, and metallic gold appearing on the metal.* Thus, for example, when powdered telluride is placed upon metallic aluminium and moistened with caustic soda decomposition immediately takes place. The same is true of zinc, although the reaction is much slower. Sodium amalgam breaks up the grains of telluride with great ease. When an electric current is used the reaction seems to be independent of the metal forming the cathode, and to take place in any neutral or alkaline solution.

"When a dilute acid was used as an electrolyte a reduction of the gold was effected with the greatest difficulty. Only by employing a very dilute acid and a large current density could any apparent decomposition be obtained.

"A possible explanation of the reaction taking place is the following: The hydrogen evolved first unites with the tellurium, forming hydrogen telluride and free gold. In the presence of alkali this compound is immediately broken up with the formation of an alkaline telluride which is easily soluble, and thus removes the tellurium from the sphere of action. This alkaline telluride is in turn easily oxidized with the formation of free alkali and metallic tellurium. In the presence of acids, however, the hydrogen telluride is immediately broken up and the free tellurium deposited *in situ*, as it were, upon the gold, thus protecting it from further action. This is indicated by the fact that a silver white piece of gold telluride becomes black when placed upon the cathode in an acid electrolyte, and also by the fact that when the amount of active hydrogen is increased by raising the current density, and the activity of the acid decreased by diluting the solution a slight deposition of gold can be obtained.

"When the rate of decomposition is rapid the gold in the small pieces of mineral is not evenly deposited, but frequently retains the original shape of the mineral fragment.

"Quantitative runs were made on a rather rich sulpho-telluride . . . using caustic soda as electrolyte, and mercury for the cathode. By maintaining good agitation it was not difficult to obtain an extraction of 93 per cent. of the assay value. The presence of finely-divided tellurium in the electrolyte seemed to cause excessive flowering of the mercury and much trouble was experienced from this cause. When lead or other solid metal was used as cathode much gold was lost by the abrasion of the moving ore.

"Good results were also obtained by agitating the ore with aluminium shavings in the presence of caustic soda solution, washing out the tellurium and amalgamating the free gold."

Reed[4] has pointed out that reduction of a metal from one of its compounds by solution of another metal, is not due to nascent hydrogen evolved by dissolving the metal added but to the electromotive force which is maintained by the energy of the dissolving metal.

I was familiar with the work of Walker and Martin a number of years before we began the Nipissing investigation and had considered the possible application of this method of reduction as a preliminary treatment for silver ores which were to be cyanided. But it had seemed to me that as the practice of fine grinding and various other refinements in

[4] Reed, C. J.: The Nascent State, *Transactions of the American Electrochemical Society*, vol. i, pp. 69 to 78 (1902).

the cyanidation of silver ores, or ores in which silver predominated, developed, extractions had reached such a high stage in the majority of cases that there was little room for the introduction of an extra operation with its attendant complication and expense, even though at the outset it were conceded that a slightly better, or possibly a more rapid, extraction could be obtained. It must be remembered that most silver minerals (there are exceptions) are soluble in cyanide solutions of proper strength when reduced to a sufficiently finely divided state and given the proper time of contact. This is even true of Nipissing low-grade ore. But with certain of the ores, excessively fine grinding is required and an almost prohibitive time of contact to recover the last of recoverable silver. Judging from a rather extensive experience in the treatment of silver by the cyanide process, together with the fragmentary and at times contradictory data available upon the solubility of silver minerals in cyanide solutions, I am forced to believe that there is a great difference in the rate of solubility of the same silver mineral occurring in different localities.

At Virginia City, Nev., the predominant silver combinations are the sulphides and sulpho-antimonides. My experience with Comstock ores covering many small-scale tests as well as actual mill operation, for a considerable period, is that these ores are favorable for cyanide treatment either direct or in combination with concentration.

The more recent experience of Symmes[5] at the Mexican mill, Virginia City, bears this out, as he has very successfully treated these ores without concentration or other preliminary treatment. This is often true of the ores from other districts in the United States and Mexico where the sulphide and sulpho-antimonide minerals of silver predominate. It is, however, true that the same degree of comminution, strength of solution in alkaline cyanide, and a period of contact is not in all cases equally effective. It is therefore evident that there is a very marked difference regarding the solubility of the sulphide and sulpho-antimonide minerals occurring in different districts, and even at times in the same mine. There are unquestionably many plants now treating ores in which the sulphide and sulpho-antimonide of silver predominate where it would not prove profitable to introduce this preliminary treatment. However, it should be realized that there might be special cases where it would be advantageous.

The use of this preliminary reduction in other districts should therefore be approached with caution and only adopted after it has been clearly demonstrated that the extraction is sufficiently improved or facilitated to produce an additional profit above the cost of the extra operation involved.

The conditions obtaining in the treatment of telluride ores are some-

[5] Symmes, Whitman: Personal communication.

what different in that on account of the *supposed* insolubility of the telluride minerals some kind of preliminary treatment is deemed necessary if the gold or silver is to be dissolved by cyanide; however, here reduction would have to compete with other forms of preliminary treatment already in successful operation.

The tellurides of gold and silver have been generally assumed to be insoluble in cyanide solutions, athough certain experimental data and mill results lead me to think that this statement is not strictly true. At any rate, even in the treatment of the lowest grade ores, some form of preliminary treatment is at present deemed essential. In the Cripple Creek district concentrating precedes or is practiced in conjunction with cyanide treatment upon the low-grade ores. Aside from these very low-grade ores where the extraction must be sacrificed for costs in order that the greatest ultimate profit may be obtained, the practice has been and still is to roast preliminary to fine grinding and cyanide treatment. It will thus be seen that the preliminary treatment of roasting for the breaking up of the telluride combination (chiefly with gold in this case) or the partially effectual operation of concentration for the removal of the telluride minerals in the case of the low-grade ore is deemed absolutely necessary for the most satisfactory results in present-day practice. However, there is no telling what the future will bring forth.

Ores containing the telluride minerals of gold and silver are not of great commercial importance in this country except in the Cripple Creek district. Therefore a consideration of possibilities in this district will be of interest. With these ores the issue would be between preliminary treatment by reduction followed by cyanide treatment as against, in the case of the low-grade ore, concentration and cyanide treatment (in most cases the concentrates are shipped to outside reduction plants); and in the case of the higher-grade ore, roasting followed by cyanide treatment.

Due to favorable location and cheap fuel the cost of roasting of the Cripple Creek ores has been brought to a very low figure and the percentage of extraction is very high upon the properly roasted ore. Of course it is only fair to say that there are losses occurring during roasting through dusting and volatilization which should be added to the cost of roasting; but in the case of the two modern mills operating at Colorado Springs these losses are doubtless small, so that in the present discussion, as they are not definitely known, they will have to be ignored. It therefore appears that the extraction would have to be fully as high by an all-wet method as when roasting was used. At the present time we have no means of knowing what it would be with telluride gold ores. The problem also arises as to whether reduction would be as effective upon the sulpho-tellurides as upon the simple tellurides. It would therefore appear that where this method has to compete with the lowest cost roasting practice in the world (dead oxidizing roast of gold ores) it might not

prove advantageous. The relative cost of a roasting plant and a plant for the wet treatment would of course be a factor which would have to be given due consideration in building new plants. If we assume that both methods are equally effective it would appear that the wet treatment plant for a given tonnage might cost less than the roasting plant, but in the case of Cripple Creek ores where the roasting plants are already installed, in view of the relative cost of the two operations, it would appear difficult for the wet process to supplant roasting in this district, unless at least as good an extraction be conceded to be possible with the wet process. However, long-range predictions are extremely difficult to make and a careful investigation of this subject in connection with Cripple Creek ores would prove most interesting and might lead to important developments.

In a district producing telluride ores where the cost of roasting was high, due to local conditions, and such a district might be discovered in the future, this process of reduction would merit particular consideration.

The data now available upon preliminary reduction are limited to its application upon a working scale to the treatment of the sulphides and sulpho-antimonides of silver at one property, and the use in this case of aluminum and caustic soda as a reducing agent. Small-scale experiments have been made upon gold tellurides in which reduction has been obtained by aluminum and caustic soda, and zinc and caustic soda, as well as by electrolysis. The effect of reduction upon other combinations and associations of the precious metals as well as an inquiry into the effectiveness and economy of other methods of obtaining reduction should be made in a thorough manner.

One question at once arises: Is the reduction selective, or in other words, what other combinations than those of the precious metals are decomposed? And in what order are they decomposed? This would seem to be a vital point, for if in certain cases it would be necessary to decompose a large proportion of valueless minerals before the reagent became effective upon the precious-metal combinations, it would certainly be a most serious drawback. There might also in certain cases be reduction products formed which would interfere with cyanide treatment or cause a high cyanide loss.

Another question which arises, particularly in regard to the sulpho-antimonides, is: What is the relative effect of using a preliminary treatment with caustic soda alone as against the use of caustic soda in combination with aluminum? There is a certain amount of evidence which seems to indicate that there are cases where all that would be necessary would be to use a preliminary treatment with caustic soda solution or caustic soda in combination with the cyanide solution.

A number of years ago, an investigation was carried on under the direction of C. W. Merrill[6] in connection with the treatment of the so-

[6] Merrill, C. W.; Personal communication.

called refractory blue ores of the Black Hills of South Dakota. In the course of this work it was discovered that a preliminary treatment with caustic soda solution in conjunction with air rendered the ore amenable to cyanide treatment. There has been at least one case in Australia where an antimonial gold ore was rendered amenable to cyanide treatment by a preliminary treatment with caustic soda. At about the same time that the Nipissing process was being developed, I had under consideration the treatment of some roasted telluride concentrate which had already been treated by the cyanide process. Realizing that perhaps the maximum effect had been obtained in the direction of an oxidizing treatment and remembering the work of Walker and Martin, I made a series of tests in which reduction treatment with aluminum and caustic soda, and zinc and caustic soda, preceded cyanide treatment. The extractions were most satisfactory, but a further series of tests in which the aluminum and zinc were omitted gave equally good extractions. It might be mentioned that the effect was not due to the presence of tellurides, for all the tellurides had been decomposed and the tellurium removed by roasting, as proved by determinations made for tellurium, but to another condition which I am not at present at liberty to enlarge upon.

Johnston states in the article under discussion that the working solutions at the Nipissing mill are more active than freshly made up solutions. Denny[7] attributes this to the presence of caustic soda in the solution, and its effect in aiding the decomposition of dyscrasite (an antimonide of silver of varying composition). It will be remembered that this mineral is not affected by the reduction treatment.

To quote further from Walker,[8] it will be noted that he greatly magnified the difficulty of applying reduction upon a large scale. It must, however, be remembered that his work was done over 10 years ago and at that time fine grinding was not generally used in connection with the treatment of gold and silver ores. In other words, the time was not ripe for the practical application of the idea.

"The enormous practical difficulties in the way of using a method of this kind is appreciated when we consider that every minute particle of mineral scattered through the relatively enormous quantity of gangue must be brought into direct contact with the surface, liberating the hydrogen before any reaction will take place. The fact that tellurides of gold are insoluble in potassium cyanide, and so far as electrolytic methods are concerned are affected only by cathodic reduction may explain why many electrolytic processes have promised well when experimenting with rich ores, but have failed when worked upon poor or lean ores. With a rich ore on a small scale it is not difficult to bring a large percentage of the individual particles into direct contact with the active surface and thus get a high extraction, even though the experimenter was

[7] Denny, James: Desulphurizing Silver Ores at Cobalt, *Mining and Scientific Press*, vol. cvii, No. 13, p. 488 (Sept. 27, 1913).

[8] Walker, W. H.: Note on the Electrometallurgy of Gold, *Transactions of the American Electrochemical Society*, vol. iv, pp. 53, 54 (1903).

ignorant of the reaction by which the values are recovered. In a lean ore, however, on a practical scale each particle of ore is surrounded by such a great quantity of gangue that an amount of agitation sufficient to keep the ore in suspension and to allow the passage of the electric current is not capable of effecting complete reduction at the cathode or other active surface."

There would seem to be several conditions which would have to be fully met to make the operation both effective and economical. First, every particle of mineral should be brought directly in contact with the reacting substances; this presents quite a problem, as the mineral particles to be decomposed form a very small proportion of the total mass which it is necessary to subject to reduction; second, the presence of any excess of the metal used in the pulp to be cyanided would cause premature precipitation and would therefore be fatal to the best results; third, the reaction would be sufficiently slow so that reduction would precede without undue waste of the reagent. To sum up: The reduction of the refractory mineral should take place as completely as possible with a minimum consumption of reagents and without any excess of the metal used remaining in the pulp at the end of the treatment, and where zinc precipitation was used caustic soda should be eliminated so far as possible.

The passage of the pulp through the tube mill containing the aluminum slugs seems to fulfill all these conditions.

The author seems to be strongly of the opinion that zinc dust could not have been used as a precipitant in the Nipissing low-grade mill on account of the fouling of the solution, as solutions containing arsenic and zinc were shown to give a diminished extraction.

Hamilton[9] is also of the same opinion, but Denny[10] is not so sure that it would not have been possible to overcome the difficulties which arose when zinc was used as a precipitant. In some of the experiments which he made in which zinc dust was used, he found that after the solution was allowed to stand eight days, it recovered its original dissolving power, but he was at a loss to explain this phenomenon. A very plausible explanation for this is apparent when the behavior of zinc precipitation in treating the high-grade ore is considered. Apparently, so far as I have been able to observe, this difficulty does not arise in treating the high-grade ore, although the amount of deleterious impurities as well as zinc passing into solution is many times that encountered with the low-grade ore. The cyanide concentration is of course much higher, which is not without its influence; but further than that the real explanation seems to be that the arsenic, antimony, etc., separate during and after precipitation from the solution to a concentration which does not interfere under the condi-

[9] Hamilton, E. M.: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal*, vol. xcv, No. 19, p. 935 (May 10, 1913).

[10] Denny, James J.: Desulphurizing Silver Ores at Cobalt, *Mining and Scientific Press*, vol. cvii, No. 13, p. 48 (Sept. 27, 1913)

tions of treatment maintained. The percentage of zinc is also reduced upon standing as well as by reactions which take place when the solution comes in contact with a fresh lot of ore and the mercury during amalgamation.

Shortly after starting the high-grade plant, I observed that there was a considerable amount of finely divided black precipitate being carried over from the foot of the precipitation boxes into the sump. This was at first thought to be a silver precipitate in suspension due to the violent action in the boxes, but it was soon demonstrated that it contained a relatively small proportion of silver. At times the solution leaving the foot of the boxes would be a clear wine color. A sample of this colored solution when allowed to stand in a glass beaker for some time became the color of the ordinary solution while at the bottom of the beaker there was deposited a black precipitate. Watson[11] gives the following analysis of the precipitate which collects in the bottom of the sumps at the high-grade mill:

	Per Cent.		Per Cent.
Silver	0.394	Iron	5.040
Mercury	2.510	Nickel	9.060
Antimony	3.300	Cobalt	7.030
Arsenic	32.640	Lime	9.240
Sulphur	16.130	Carbon dioxide	7.259
Silica	5.362	Manganese	trace
Zinc	2.257		

It is evident that these impurities are much more difficult to precipitate than the silver, for they seem to separate largely at the foot of the precipitation boxes, and indeed a part of it goes over as a colloidal solution from which the precipitate separates completely only after standing several days in the sump tanks. It might be mentioned that agitation seemed to facilitate this separation. In precipitating the solutions from the first large-scale tests by zinc-shaving precipitation, the wine-colored colloidal solution was noted at times at the foot of the precipitation boxes. However, in general, it was not apparent with the low-grade ore on account of the much smaller percentage of the deleterious elements which it contains. In general zinc-shaving precipitation throughout my tests gave very complete and statisfactory precipitation; but it must be remembered that conditions were somewhat different than those obtaining in the treatment later developed. My own observation, together with facts given by Denny, led me to think that zinc precipitation under proper conditions would not have been impossible in the cynidation of Nipissing low-grade ore. However, it would, no doubt, have required a very careful investigation of the conditions governing the separation of the undesirable impurities from solution during and after precipitation.

[11] Watson, R. B.: Nipissing High Grade Mill, *Engineering and Mining Journal*, vol. xciv, No. 23, p. 1080 (Dec. 7, 1912).

Perhaps the chief interest in the discussion of this feature of Nipissing practice centers about its applicability elsewhere. This is a subject which must be approached with considerable caution, for, despite the general prejudice which has existed against zinc since the cyanide process was first introduced, zinc has in the great majority of cases been giving most excellent results, particularly when used in the form of zinc dust. This statement is made as a result of a most critical consideration of the whole field of precipitation with a view to finding a substitute for zinc which would perform all its functions. The use of aluminum or even aluminum dust as a precipitant is by no means a new idea, so that if it did not possess some very obvious disadvantages it would have come into general use as a precipitant for gold and silver from cyanide solutions long ere this. I am inclined to think that its future use will be still confined to special cases.

Moldenhauer[12] first proposed the use of aluminum as a precipitant and obtained a patent for its use in 1893.

Early experiments with it by Julian[13] in South Africa in the form of plates and shavings with solutions containing gold, did not prove satisfactory. The following quotation from Julian will give an idea of the difficulties encountered:

"It is necessary for efficient precipitation that either free alkali or free acid should be present in the solution. For obvious reasons, the latter is difficult to apply, if not impracticable, in the cyanide process. It was found after the process had been at work some weeks that aluminum began to deposit on the shaving, which retarded precipitation of the gold. This was easily rectified by the addition of a little alkali at the head of the box, but as time went on the ever increasing quantity of alumina in the working solutions became more and more troublesome. Attempts were made to remove the alumina by precipitating it in the solution tanks, and then leave it behind in the ore, but it was found that *whenever alumina was precipitated in a cyanide solution it also carried down a large proportion of the free cyanide.* Methods have been suggested for overcoming the difficulty, but at a prohibitive cost."

Kirkpatrick[14] began experiments in 1906 with aluminum dust as a precipitant and states that, when used in the form of dust, the difficulties experienced by the earlier experimenters are overcome. Nevertheless, it is difficult to see how this could have any effect upon the formation and separation of alumina from the solutions. The use of aluminum dust was first applied commercially by the Deloro Mining & Reduction Co. in 1908 and was in use as late as June, 1913, in connection with the cyanidation of the speiss resulting from the smelting of Cobalt high-grade ores. The same interests introduced its use at the O'Brien mill, in the Cobalt

[12] Moldenhauer, Carl: New Zealand patent, Aug. 31, 1893.

[13] Julian, H. Forbes, and Smart, Edgar: *Cyaniding Gold and Silver Ores*, 2d ed., pp. 164 and 165 (London, 1907).

[14] Kirkpatrick, S. F.: Aluminum Precipitation at Deloro, Canada, *Engineering and Mining Journal*, vol. xcv, No. 26, p. 1277 (June 28, 1913).

district, where, although in use about three years, it attracted but little attention. The general character of the ore treated at the O'Brien mill is similar to that treated at the Nipissing low-grade mill. Rough concentration is practiced prior to cyanidation.

The high cost of aluminum dust as compared with the cost of the metal in other forms, as well as a part of the difficulty of wetting it and the comparatively long period of violent agitation necessary even in precipitating silver solutions, due to a coating of the oxide and stearine, is largely explained by a consideration of the method of manufacture.[15]

It is necessary to start with the foil or leaf as a raw material. This is manufactured by a special system of rolling or combined rolling and hammering, and it is about 0.01 mm. in thickness. With either process, the percentage of first-class foil is only 33 to 35 per cent. of the 0.04-mm. metal started with. The imperfect foil is converted into dust by comminution in two series of special stamp mills. The product from the first series of stamps is classified and the finest of the product separated and passed on to the second series of stamps for the completion of the operation. During this second stamping, the difficulty due to the agglomeration and welding of the particles is overcome by the addition of 2 *per cent. of stearine.* The product of the second series of stamps, after being screened through No. 200 silk bolting cloth, is further classified by a winnowing process. Although the finished product is polished in a special cylinder by brushes, a considerable proportion of the 2 per cent. of stearine used to prevent agglomeration must still remain as a coating upon the particles of the finished product.

Butters[16] has sought to overcome some of the difficulties of aluminum-dust precipitation by a system employing granulated aluminum in a tube mill through which the solution to be precipitated is passed during rotation of the mill. The bulk of the precipitate is to be collected by subsequently passing the solution through a filter press.

In order to come to the point, let us ask a few pertinent questions: First, is aluminum-dust precipitation cheaper than zinc-dust precipitation? (Of course in this connection we must give due consideration to all factors, even those having an indirect bearing upon the subject.) Second, is there any function which it would not perform which zinc does? Third, is it as effective for dilute solutions containing a small proportion of metal as is often encountered in treating low-grade gold ores?

The average consumption of aluminum dust over a period of nine months is given as 1 lb. avoirdupois for each 3.104 lb. avoirdupois of silver precipitated. In the Pachuca district[17] it has been regular practice with a Merrill zinc-dust precipitation system to precipitate 1 lb. of

[15] Process used at La Praz and Charleval-sur-Andelle by the Société française de Couleurs métalliques. L. Guillet: *Revue de Métallurgie,* vol. ix, pp. 147 to 159 (1912); also *Mineral Industry,* vol. xxi, pp. 23, 24 (1912).

[16] U. S. patent No. 1,092,765, Apr. 7, 1914.

[17] Merrill, C. W.: Personal communication.

fine bullion per pound of zinc dust, and with a double-circuit system of precipitation even better results have been attained. There is no doubt that this result could be duplicated or bettered at the Nipissing low-grade mill, for in general, with zinc precipitation, the greater the proportion of silver in the solution to be precipitated, other conditions being the same, the higher the efficiency.

Hamilton[18] gives the cost of zinc dust laid down at the Nipissing plant at 7c. per pound while aluminum dust costs 35 to 39c. per pound. Suppose that no charge is made for the caustic soda necessary with aluminum or the extra power necessary for agitation, or no allowance for the cyanide carried down with the alumina or calcium aluminate which sooner or later must separate from the solution, and no credit is given the aluminum for cyanide regenerated, we find that the actual cost of precipitating a pound of silver with aluminum would be 11.3 to 12.5c., while with zinc dust the cost of precipitating a pound of silver would be 7c. Upon this hypothetical basis, the costs are very much in favor of zinc. The cyanide regenerated has of course a most important bearing upon the cost of precipitation as the value of all that is actually saved should be deducted from the total cost of aluminum precipitation. I prefer to use the term saved, rather than regenerated, for the reason that cyanide may appear to be regenerated, as indicated by titration at various stages of the process, and still this may not constitute a real saving in the amount eventually used.

The regeneration of cyanide noted is 1.67 lb. per ton or 408 lb. per day. And as this is given as one of the practical benefits gained by a change in the mill plan from zinc-dust to aluminum-dust precipitation, it deserves careful consideration. (Zinc dust was never used in the completed mill.) In other words, it is claimed that there would have been required 1.67 lb. per ton of ore treated or a total of 408 lb. per day additional cyanide if zinc precipitation had been used.

This claim for cyanide regenerated is evidently based upon the difference between titrations made upon the solutions before and after precipitation. There will be considerable difference of opinion regarding just how much weight should be given determinations of this kind as indicating the amount of cyanide which would actually be saved to a plant through aluminum or any other form of precipitation. As having a direct bearing upon this question, the extreme view of Hamilton[19] is quoted:

"Some may dispute the ground I take when I count as lost the cyanide that remains combined with zinc, because it is often stated that the double cyanide of zinc and potassium is almost as efficient for dissolving purposes as the simple cyanide. My experience is, however (at any rate in the case of silver ores), that the reading obtained by the use of potassium iodide indicator with excess of caustic, is worthless as a

[18] Hamilton, E. M.: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal*, vol. xcv, No. 19, p. 939 (May 10, 1913).

[19] *Idem*, p. 936.

measure of the dissolving power of a cyanide solution, the efficiency being for practical purposes proportional to the 'free' cyanide reading obtained by stopping at the first faint opalescence without the use of potassium iodide indicator."

Now if we concede this saving in cyanide, claimed through precipitation with aluminum, it is necessary for us to agree with Hamilton's statement regarding the absolute loss of all cyanide combined with zinc, and concede the entire effectiveness of that liberated by aluminum.

Most operators will disagree with this extreme view, but regarding the degree to which cyanide combined with zinc becomes, later, available for extraction there would of course be a great diversity of opinion.

There is no accurate method for the determination of the uncombined alkaline cyanides in mill solutions. The method invariably used in present-day mill work is the well-known Liebig titration, which employs a solution of silver nitrate. Practice in its application differs somewhat in that at times certain constituents in the cyanide solution are depended upon to act as an indicator, and again an independent indicator is added. A great variety of results will be obtained from the same mill solution, particularly if much zinc is present, depending upon the indicator used and the extent to which the solution has been diluted prior to titration. The method of titration without dilution and without an indicator generally gives the lowest result and perhaps the one which, in most cases, is the most nearly a measure of the simple alkaline cyanide present, but let no man delude himself into thinking that it gives an accurate result in a mill solution, especially one containing zinc, whereas the other methods are inaccurate. This procedure, while relatively more accurate except in the presence of copper than that in which potassium iodide (the most generally used indicator) is added and the titration carried to the first yellow opalescence instead of stopping at the first faint white opalescence, as in the method when an indicator is not used, is more difficult to carry out. Its proper performance requires considerable experience, absolutely clean apparatus, and a good light. As the cyanide indicated by any modification of the Liebig method is only relative, a great many operators prefer to use potassium iodide as an indicator on account of the greater ease with which the determination is made. If proper allowance be made for the higher result obtained when potassium iodide is used, there is no reason why the operation of a cyanide plant cannot be just as successfully carried on when guided by titrations made with potassium iodide as an indicator as by the other method. In fact, where the ordinary solution man is depended upon to make the titrations, as is usually the case, there is less chance of mistake. *The important point to bear in mind is that in any particular mill the same method should be adhered to throughout so that results are comparable.* A good illustration of this is one which came under my observation of a mine manager who, having made a record for low costs at one mine, was appointed manager of an adjoining mine. In the course of time, the

cyanide plant came in for its share of consideration in the operation of cost shaving, and it required only one glance to satisfy him that too much cyanide was being used, for the solutions all through the mill were being kept at a higher cyanide strength than had been found necessary at the mine he had just left, which was in the same district and had the same general character of ore. He, therefore, ordered that no more cyanide be added until the strength of the solutions fell to the level which his former experience had shown was most advantageous. Remonstrance upon the part of the cyanide chemist was of no avail, for this mine manager was supreme, and above the details of cyanide titrations, and had a record for low costs to maintain. The effect was apparent in a few days, and in as many weeks the situation became critical and the mine manager was frantically calling for help, for although he was reducing costs upon cyanide consumed he was not obtaining the extraction, and he saw that the end was at hand unless something heroic was done at once. Finally, a chemist at his former plant came to the rescue. Being a wise chemist, and having had considerable previous experience in cyanide plants, he at once titrated the solutions without potassium iodide indicator, and to his surprise found that they contained little free cyanide. Knowing the remedy, he at once began adding cyanide in a most reckless manner and extractions began to improve and were soon back to the old level. The solution strength was maintained the same as in the manager's first plant, the titration employing potassium iodide was discarded, and with it went the former chemist. The manager was happy for he was operating with the same solution strength used in his former plant, and extractions were as good as when he came to the plant. *But there was no saving in the cyanide consumption.*

I was once called upon to investigate the treatment at a mill where potassium iodide was used as an indicator in making the cyanide titrations. In making my tests, the titration without an indicator was used. In the course of time, it was determined that a mill solution of about 0.10 to 0.11 as indicated by this titration was the best, everything considered, for treating the slime. The solutions in the plant as daily recorded by the solution men, who used potassium iodide as an indicator, ranged from 0.14 to 0.15 per cent. I was rather surprised when I came to titrate these same solutions to find that they ran 0.10 to 0.11 per cent. as indicated by the method which I was using. This shows that two independent investigations of the treatment of this ore had produced the same result, although a different method of titration had been used by the man who first started the plant and the absolute indications by this method were higher than by the method I used, which was the same as that so strongly advocated by Hamilton.

Hamilton's wholesale condemnation of the use of potassium iodide as an indicator is the result of making too rigid comparisons between titra-

tions made with and without its use. This is proved by the fact that the operation of many cyanide plants is controlled with entire satisfaction by the Liebig titration, employing potassium iodide as an indicator.

Having considered the methods of determining the simple alkaline cyanides, we can proceed with a consideration of the extent to which potassium zinc cyanide is available as a solvent for gold and silver. The following is quoted from Sharwood,[20] who has made a critical investigation of the properties and behavior of potassium and sodium zinc cyanides:

"*The considerable increase in solvent power observed when caustic potash is added* (always providing that oxygen is accessible) *coupled with the superior effect of free potassium cyanide, is good evidence that some free potassium cyanide is formed upon such addition,* and therefore that in dilute solutions *potassium zinc cyanide is partially decomposed by caustic alkali, with formation of simple alkaline cyanide,* in accordance with the principle of chemical equilibrium between substances in solution."

Williams[21] states that the free cyanide shown by titration to opalescence without potassium iodide is, within limits, proportional to the amount of alkali present. Williams[22] also states that lime (CaO) when added in solution to the double cyanide of zinc and potassium has the same effect as potassium hydrate per equivalent added, with the exception that the zinc is precipitated as hydrated zinc oxide when silver nitrate is added to the solution.

Virgoe[23] gives comparative results showing the extractive power of potassium zinc cyanide, plain cyanide, and potassium zinc cyanide with the addition of lime. He[24] comments upon certain of these results as follows:

"The value of the addition of the alkali is obvious, and this salt is an excellent solvent for gold, though comparatively not so good a solvent for silver, even in the presence of lime, though this may be simply a question of duration of treatment."

Again he[25] comments upon the results given in Table 2.

"Thus it is seen how excellent a solvent this salt is in certain cases, and how the extractions are accompanied with a much lower cyanide consumption."

The regeneration by the addition of alkaline hydroxides of simple alkaline cyanides which dissolve gold and silver, has been demonstrated

[20] Sharwood, W. J.: A Study of the Double Cyanides of Zinc with Potassium and with Sodium, *Journal of the American Chemical Society*, vol. xxv, pp. 580, 581 (1913).

[21] Williams, Gerard: The Determination of Constants in Working Cyanide Solutions, *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 491 (1903–1904).

[22] *Ibid*, p. 487.

[23] Virgoe, Walter H.: Consumption of Zinc in Cyanide Plants, Nature, Cause and Effect, *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 630 (1903–1904).

[24] *Ibid*, p. 629.

[25] *Ibid*, p. 633.

upon many occasions in mill practice by raising the alkalinity of the solutions and running for short periods without the addition of cyanide or with the addition of a much smaller amount of cyanide than that normally necessary.

A good illustration of this was the announcement made a number of years ago by a mill superintendent, treating a silver-gold ore, of the discovery that cyanide could be regenerated by running with a high alkalinity. This proved effective for some time in reducing the amount of cyanide used, but later, after the reserve of cyanide carried with the zinc had been largely depleted, it was found necessary to return to the old proportions of cyanide. In the ordinary mill solution, when operated under normal conditions, the proportion of zinc in solution remains more or less constant, so that all zinc taken into solution through precipitation above this maximum, which is reached shortly after the plant begins operation, is separated out as insoluble compounds through reactions, at least a portion of which result in regeneration of cyanide effective for the solution of gold and silver.

With aluminum precipitation, it is necessary to add caustic soda just prior to precipitation, and as aluminum does not even temporarily combine with cyanogen, all the cyanide regenerated is at once apparent by the ordinary Liebig titration without an indicator.

With zinc precipitation, the addition of an alkali to the solution prior to precipitation is not necessary and in the great majority of cases would be a distinct disadvantage, as it would result in a greatly increased solution of zinc with its attendant disadvantages. However, suppose we add caustic soda to the solution, after zinc precipitation, and then make titrations as before, we find that there has been a considerable regeneration of cyanide, for reasons already made sufficiently clear. Now this is just what happens in practice as the alkali is added after precipitation, or generally with the ore in the form of lime, and therefore comes in contact with the solution as each fresh charge of ore comes under treatment. During treatment, regeneration is going on to a greater or less extent during the whole period. The rather slow regeneration which takes place during treatment when zinc precipitation is used is not without its advantages, as the cyanide is gradually freed from the combination, as needed. That this makes for a somewhat lower cyanide consumption is proved by the general experience of there being a less cyanide consumption in regular mill practice than when starting a new mill when there is no zinc in solution, or as indicated by small-scale tests in which fresh solution is used.

There are perhaps other causes which contribute to this, but this factor undoubtedly accounts for a large part of the difference. Regeneration taking place during treatment is not apparent by titration, as decomposition of cyanide is going on at this period at a more rapid rate

than regeneration. While there are other reactions taking place during treatment which cause regeneration, such as dilution, etc., yet the principal influence appears to be the addition of alkali. With zinc precipitation it is therefore apparent that the various reactions which cause regeneration only take place when the solution is applied to the treatment of a fresh lot of ore when zinc separates and cyanide becomes available for extraction, while with aluminum precipitation all the factors contributing to regeneration are present at the moment of precipitation.

Unfortunately, this indirect regeneration as we may term it, is not readily capable of direct quantitative determination as in the case of aluminum precipitation, so that the only way to make direct comparisons of the regeneration taking place in each case, would be to run the same plant upon the same ore for separate periods of time of considerable length, employing both forms of precipitation. A comparison between the actual amount of cyanide consumed in each case would tell the tale. When one has all the evidence before him, he is compelled to admit that there is regeneration of cyanide at certain stages of the process when zinc precipitation is employed, and remembering Julian's statement that alumina—and this and the aluminate are the forms in which aluminum separates from cyanide solutions—carries down with it cyanide, it is very evident that the saving in cyanide at the Nipissing mill, through the use of aluminum precipitation, is not so great as has been claimed.

Zinc, in solution, is not without its advantages, as it serves a useful function in removing soluble sulphides and to a certain extent in acting as a protector for the cyanide temporarily combined with it. An example of the important function which zinc at times may perform in cyanide solutions is the experience of Colbath[26] when treating ore from a certain stope in the El Rayo mine. The extraction decreased to a considerable extent, and trouble arose with precipitation. Coincident with these difficulties, it was noted that the zinc had disappeared from the solutions to the extent that the titration for total cyanide coincided with that for simple cyanide. Lead acetate was tried as a remedy, but it did not prove effective, so the rather startling experiment of adding zinc in the form of potassium zinc cyanide was tried.

This addition of zinc caused the extraction and precipitation to again become normal. This unusual case, where there was not zinc enough supplied to the solutions through precipitation, very forcibly draws attention to the important function which zinc stands ready to perform. Ordinarily, sufficient zinc is supplied to the solutions through solution of zinc during precipitation. Evidently, in certain cases, aluminum precipitation could not perform all the functions that zinc precipitation does.

[26] Colbath, James: Personal communication.

It seems to me that the process of reduction and aluminum precipitation work remarkably well together under the Nipissing conditions, for the possible disadvantage of caustic soda carried over from the reduction process, if zinc precipitation were used, becomes an advantage with aluminum precipitation. The caustic soda thus carried over and that added during precipitation on account of antimonal compounds, is a decided aid to the extraction of the silver. The preliminary reduction treatment presumably removes compounds which would tend to form soluble sulphides, so that the non-effectiveness of aluminum for the purpose of removing them is perhaps of little moment in this particular case.

Discussion

Thomas Crowe,* Victor, Colo.—The interest manifested of late in the treatment of low-grade ores, together with Mr. Clevenger's discussion of The Mill and Metallurgical Practice of the Nipissing Mining Co., prompts me to add a few remarks relative to concentration in connection with cyanide treatment of low-grade ores.

Mr. Clevenger in this discussion does not condemn concentration in this connection, but, nevertheless, the tone of his remarks would lead one to believe that his conclusions are like those of many others: that concentration is often turned to as a last resort in an attempt to improve or obtain an extraction upon an ore by the recovery and sale of the refractory portion of the ore. This, I will attempt to point out, is not always the case. Concentration in connection with cyanidation often performs an entirely different function, *i.e.*, one of saving fine grinding.

Economy being the keynote of successful treatment of low-grade ores, the problem often becomes more commercial than metallurgical, and, as there is generally a definite ratio existing between cost of operation, degree of comminution, and percentage of extraction, the grade of ore under treatment usually imposes a limit upon these factors.

With many ores grinding is the most expensive single item in their treatment; therefore, the degree of comminution is very apt to be governed by the allowable cost of operation. With most ores the degree of comminution controls to a great extent the percentage of extraction. So in the treatment of low-grade ore it often becomes necessary to sacrifice extraction, through coarse grinding for the benefit of cost, in order that the greatest ultimate profit may be obtained, and it is under these conditions that it is possible for concentration to play an important part in overcoming to some extent the effect of mesh.

The precious metals occurring in an ore are usually closely associated

* Non-member.

with the metallic portion of the ore, and as this metallic portion is generally fairly well liberated from the gangue at comparatively coarse meshes, further grinding of the ore is necessary only in order that the metallic portion may be reduced sufficiently fine that the precious metal part of it may be dissolved by cyanide solutions in a reasonable length of time.

A concentrating table under these conditions would have the effect of removing this refractory metallic portion, it being especially efficient in removing that portion which is not sufficiently fine to be readily dissolved by cyanide solution (the coarse), putting the small amount of high-grade concentrate in a separate pile where it can be dealt with by more extensive methods of treatment, it being of sufficient value per ton to justify further grinding, longer contact, and more elaborate methods, at the same time simplifying the subsequent treatment of the bulk of the ore and accomplishing the same result as though the whole mass of ore were ground to a very fine mesh.

An exemplification of the effect of concentration in connection with cyanidation in the treatment of low-grade ores may be found in the mills of the Cripple Creek district, which are treating the sulpho-telluride dump ores. Here concentration performs another function besides that of saving grinding as described above. On account of the peculiar occurrence of the values in these ores, the sulpho-tellurides occurring upon the faces and seams of the rock, when the ore is crushed to 30 mesh it is found that the sulpho-tellurides are liberated to such an extent that, after concentration and classification, the sand product of this operation is of such low value that it can be rejected as a tailing, leaving only the enriched concentrate and slime to receive further treatment.

The low treatment costs allowable by this rejection of 50 per cent. of the ore in the form of low-grade sand can be well imagined. In fact, the success of these mills in the treatment of this low-grade by-product is only made possible through the continual elimination of that material which will not withstand further treatment. This is practiced by other methods of concentration besides table concentration, such as hand sorting, coarse crushing and trommeling, etc., and I would like to make a long-range prediction that in the future low-grade milling selective methods will prevail and concentration become an important factor.

G. H. CLEVENGER, Palo Alto, Cal.—The whole question of ore treatment is, of course, an economic one and frequently our pet metallurgical theories have to be sacrificed upon the altar of greatest ultimate profit. If the recovery of a portion of the gold and silver can be more economically made by concentration than by solution in cyanide, obviously concentration should be practiced. The case cited by Mr. Crowe is an unusual one in that his strongest argument for concentration is that the

sand can be rejected without further treatment if the concentrate is removed. It must be remembered, in this connection, that the ore treated runs less than $3 per ton and that a large proportion of the minerals carrying the gold occur along the cleavage planes. Even with the same character of ore, if of considerably higher grade, it would not be possible to reject the sand without incurring a serious loss. The character of many of the low-grade ores of other districts would render this type of practice impossible. Lack of suitable mill sites for a large expanse of leaching tanks, and favorable smelter contracts, are factors in the Cripple Creek district which are not without their influence.

There are a number of possible variations of concentration in conjunction with cyanidation, the more important of which are:

1. Crushing of the ore in water; concentration, either directly or following another recovery operation, as amalgamation; rejection of the tailing, and cyanidation of the concentrate. This method is most suitable for use upon very low-grade ore or tailing. A good example of such practice is the Treadwell, where the tailing from the amalgamation of the very low-grade ore treated could not be profitably treated directly by cyanidation; but cyanidation of the concentrate recovered from the tailing by concentration returns a handsome profit.

2. Crushing of the ore in cyanide solution; concentration, followed by cyanidation of the sand and slime. Concentrate treated by one of three methods:

(*a*) Shipment of the concentrate to the smelter.

(*b*) Special local treatment of the concentrate.

(*c*) Special treatment of the concentrate stream, as, for example, fine grinding or amalgamation, etc., and return of the concentrate stream to the balance of the pulp for cyanidation.

Method (*a*) of concentrate disposal has been very generally practiced in the past, but care must be exercised in adopting this practice, for the reason, as I have previously pointed out, that one may pay the smelter rather dearly for recovering gold and silver recoverable by cyanidation under proper conditions.

Method (*b*), involving special chemical treatment or roasting prior to cyanidation of concentrate, has been practiced and, under certain conditions, may be advantageous.

Method (*c*) has its adherents and, under favorable conditions, may present certain advantages. However, there is at present a tendency to revert from this method to (*b*), even when it is possible to obtain as high a recovery by (*c*), for the reason that, if the concentrate residue is kept separate, it may later become a valuable asset.

3. Crushing of the ore in water or cyanide solution and separation of the pulp into sand and slime; concentration of either one or both; rejection of either sand or slime and cyanidation of the other product.

An example of this is the old Homestake practice, where, after crushing in water and amalgamation, the slime was rejected, as it was too low grade for profitable treatment until the development of the Merrill filter press. The sand in this case was treated by cyanidation without concentration. At the Portland Victor mill the sand is rejected after concentration and the slime treated by cyanidation.

ALLAN J. CLARK, Lead, S. D.—A point that might be made against aluminum precipitation is that the necessity for free alkali in some quantity would prohibit its use in connection with the treatment of ores where a very low protective alkalinity was necessary for the best results regarding either extraction or cyanide consumption. A good example of this is present Homestake practice, where an extremely low protective alkalinity has been found advantageous.

G. H. CLEVENGER.—Mr. Clark's experience with Homestake ore has also been borne out by my own with certain silver-gold ores, where the best results were obtained with a low alkalinity. However, there are exceptions to this.

Chloridizing Leaching at Park City

BY THEODORE P. HOLT, PARK CITY, UTAH

(Salt Lake Meeting, August, 1914)

Outline of the Process

THE Mines Operating Co.'s plant at Park City, Utah, was designed to treat the low-grade fillings in the old stopes of the Ontario mine. These fillings carry 6 to 14 oz. of silver, 1 to 2 lb. of copper, 0.01 to 0.015 oz. of gold, and a small percentage of lead and zinc. The treatment consists in mixing the crushed ore with coal dust and salt and then roasting in a new type of furnace by combustion of the contained fuel. The roasted ore is leached with an acid salt solution to dissolve the silver, gold, copper, and lead. At present these metals are precipitated together on scrap iron and the product sold to a refinery.

The only new feature of importance in this scheme of treatment is the roasting process, which makes possible the chloridizing of ores without any loss of the valuable metals, and at a very low cost.

History of the Development of the Process

The roasting process had its beginning in connection with research work at the Utah State School of Mines. N. C. Christensen, Jr., who held one of the School of Mines' fellowships, was engaged in some experimental work on blast roasting. At the same time I was doing some work on the insoluble gold in Mercur base ores. Mr. Christensen roasted some of this base gold ore in his pot furnace, and found that when he mixed a sufficiently low percentage of fuel with the ore, it did not sinter but roasted to a leachable product.

We considered this a new application of blast roasting, and proceeded to test out a large number of ores in several different types of small roasters which we constructed. This experimental work continued well into the summer of 1911.

In August, 1911, the Consolidated Mercur Gold Mines Co. installed a small roaster of our design, which we proposed to operate on the countercurrent principle, feeding mixed ore continuously on top, and at the same time drawing off the roasted ore from the bottom. By proper adjustment we hoped to maintain a permanent roasting zone near the center of the column, the ore moving downward as roasted. However, we

experienced difficulty in getting the moving mass to roast properly, and also other troubles of a mechanical nature developed. Mr. Christensen at this time disposed of his interest in the process to G. H. Dern.

We continued experimental work on a number of machines of varying design. Up to the time construction work began on the Mines Operating Co. mill no satisfactory continuous roaster had been developed, so we decided to install an intermittent roaster, which our experience had demonstrated would give a satisfactory product for subsequent leaching operations. Accordingly, we installed roasters similar to the one shown in Fig. 2, the construction and operation of which will be described in its proper place in the description of the plant.

The Plant

The Mines Operating Co.'s plant is installed in the old concentrator of the Ontario Silver Mining Co. An outline of the present scheme of treatment may be followed by reference to the accompanying flow sheet (Fig. 1).

A second-motion electric hoist, operating 1.2-ton skips in balance, delivers the ore from the haulage tunnel of the mine to a bin above the crusher. From here it is fed by a Stephens-Adamson apron feeder to a trommel with 1½-in. openings. The undersize of the trommel passes through a revolving drier, where the moisture is reduced to about 5 per cent. The oversize of the trommel falls upon a picking belt, where mine wood and waste are removed. It then passes through a No. 5 Gates gyratory crusher and, joining the drier fines, is elevated to the crushed-ore storage bin. This bin has a capacity of 400 tons of ore. Sections of the same bin are respectively used for salt and coal storage. From these bins the ore, salt, and coal are fed, in the required proportions, upon a belt conveyor and delivered to the rolls for final crushing. The feeders by means of which the proportioning and mixing are done are of the plunger type, the feed being varied by changing the eccentric throw. This type of feeder works very well for the salt and coal dust, but is not very satisfactory for ore, on account of rapid wear. The mixed ore is crushed in two sets of 15 by 36 in. rolls and is elevated to the mixed-ore bins above the roasters.

There are eight shaft roasters of the intermittent type, similar to the one shown in Fig. 2. The roaster shown is 10 by 10 ft. inside the walls. The walls are reinforced concrete 10 in. thick. About 2 ft. above the bottom is a wood grating, supporting a layer of coarsely crushed rock. The space below this grate forms an air chamber. The grate supports the roasting charge, and also distributes the air blast.

The method of operating the roaster is as follows: A special starting "mix" of about 1 ton is prepared in front of the roaster. A layer of coal

dust mixed with oil is then spread over the gravel floor of the roaster. This is ignited and sufficient blast admitted to burn it rapidly to glowing coals. The special mix is then shoveled evenly over the surface. By the aid of the air blast, the coals ignite the fuel in the ore, which begins to roast. When the starting layer has roasted through, so that the fire appears on top, the air is shut off for a few minutes, while the first charge

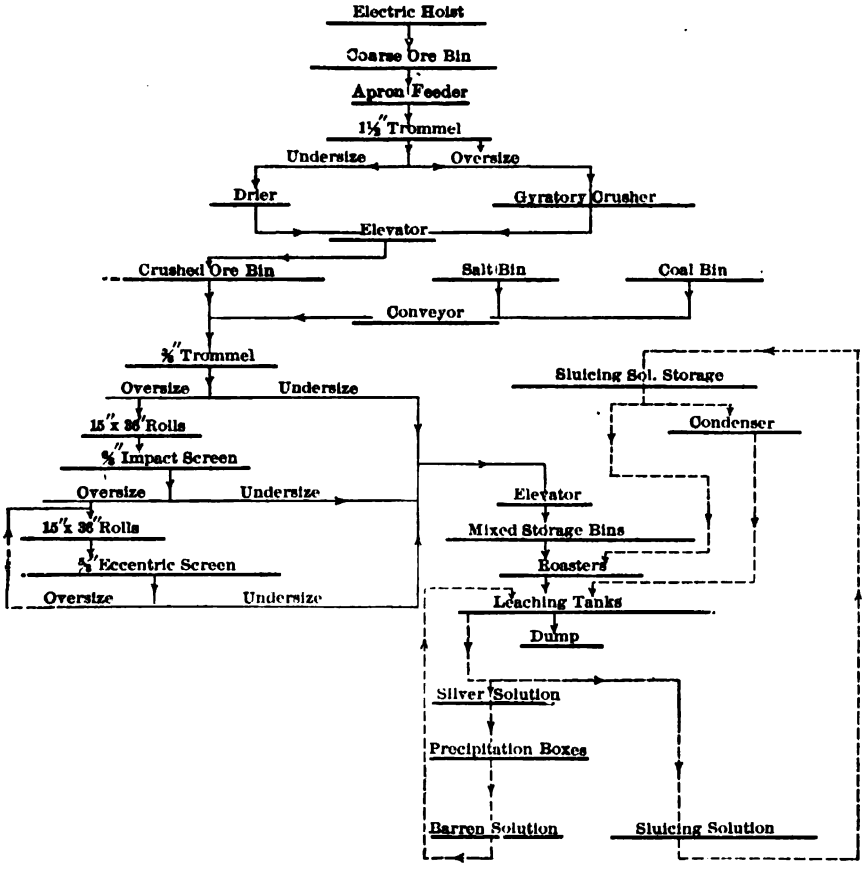


FIG. 1.—FLOW SHEET OF MINES OPERATING CO.'S MILL, PARK CITY, UTAH.

of about 5 tons is dropped from the mixed bin gate into the roaster. This is spread out by hand and the air again turned on. The roast is allowed to proceed until it appears on the surface of the charge. Then a second charge of about 10 tons is dropped on and 4 hr. later a final charge of about the same amount. This brings the total depth of charge up to about 7 ft. Under normal conditions this will roast through in 8 to 12 hr.

When the roasting zone has reached the surface at all points the air blast is shut off. The discharge door is opened and a sluicing apron inserted, to connect with the launders. The hot roasted ore is then sluiced out with mill solution. The sluicing nozzle is made of hard wood and has a $1\frac{1}{8}$-in. nozzle opening. It operates under a head of 45 ft. The nozzle is suspended in position in front of the discharge door. The operator stands at one side, to avoid the numerous steam explosions, and directs the nozzle with a long rod. Under normal conditions it takes from 1 to 3 hr. to sluice out a roaster charge of 24 tons.

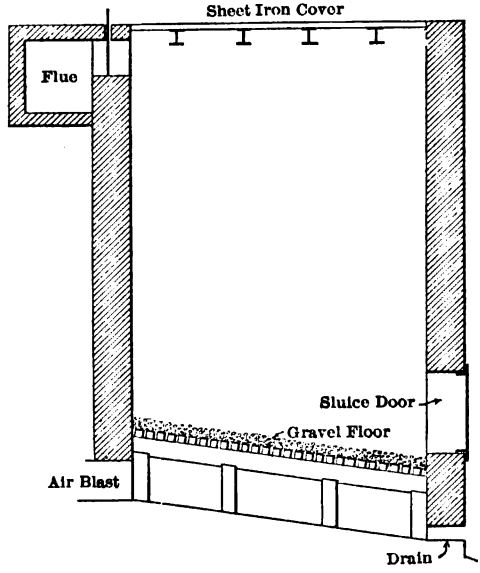


FIG. 2.—SHAFT ROASTER.

From the roasters the hot ore mixed with mill solution passes by means of launders to the leaching tanks. The leaching tanks are 20 ft. in diameter and 12 ft. deep. There are six of these, each holding 135 tons of ore. In the bottom of each tank is a 12-in. round discharge hole for sluicing out the tailing. This is closed by a turned wood plug, having a 4 by 4 in. stem extending to the top of the tank. For convenience in sluicing out, the discharge hole is surrounded by a box extending to the top of the tank and provided at intervals with 12 by 12 in. sluice gates.

Of the various filter bottoms tried, one made of 4-in. drain tile has given the best service. Four lines of tile are spaced across the bottom of the tank, and cemented in position by a thin slab of concrete on each side. This tile system connects at one side of the tank with a 3-in. wood-pipe solution line.

The solution lines from the leaching vats lead to a distributing box.

Here the solution is directed either to the "weak" tank or the "silver" tank, depending upon its metal content. From the weak storage tank the solution is pumped back for sluicing purposes, while from the silver tank it goes to the precipitating boxes.

The iron precipitation boxes are similar in design to the wood zinc boxes sometimes used in cyanide mills. They are larger, however, and so constructed that the iron tie rods are not in contact with the acid-soaked wood. From the precipitation boxes the solution passes to the barren sump and is returned as a leaching solution to the vats.

New Features

Having followed the ore treatment through in a general way, we will now return to discuss more in detail certain features that are more or less new.

Importance of Proper Mixing

Perhaps the most important step in the whole operation is the proper mixing of the ore preparatory to roasting. Roasting for lixiviation requires a very close regulation of temperature. The best results on many ores are obtained between 600° and 700° C. In any case, to sinter parts of the ore is to render it unfit for subsequent lixiviation. The three chief factors which determine the temperature are: (1) the percentage of fuel in the mixture, (2) the percentage of moisture in the mixture, and (3) the amount of air blown through the charge per unit of time. The first of these three is much the most important.

Coal dust is used as fuel. The present supply costs 25c. per ton f.o.b. cars at the coal mine. Since this class of material takes a cheap freight rate it is the most economical fuel to use under our conditions. The amount required varies from 2.4 to 3.0 per cent. of the weight of the ore being roasted.

The ore as it comes from the mine is very wet. The drier is supposed to reduce the moisture to about 5 per cent., but the variation is considerable. We might suppose that an increase in moisture would lower the temperature of the roast. It has exactly the opposite effect, however, and it is necessary to reduce the percentage of fuel as the moisture in the ore increases.

The mill was designed to treat 150 tons per day. It is handling considerably more than this, the average for the month of March being 166 tons. This tonnage is mixed and rolled on one shift. The mixing is in charge of a trustworthy man who checks up the feeders by weighing the output of each frequently.

The New Holt-Dern Roaster

During 1913, a new continuous roaster was developed, which overcomes the difficulties experienced with previous machines. A full description of the roaster will not be attempted in the present paper. A few of its general characteristics and the results obtained will be noted.

We installed a commercial-size machine at our plant in December, 1913. Except for a few brief delays for changes, it has been in continuous operation. In the Holt and Dern roaster the column of ore moves down at intervals as the roasting zone travels upward. The air blast travels in an opposite direction to the ore. In this way the air passes first up through the hot roasted ore and becomes highly heated. It then passes through the roasting zone where active combustion of the fuel in the ore is taking place, and finally through a layer of moist unroasted ore. Hence when it leaves the roaster it is fairly cool and entirely free from dust.

Our experience at this plant has demonstrated the following advantages in this roaster over the intermittent type already described:

Comparative Cost Data per Ton of Ore Roasted

	Old Roaster	Holt & Dern Roaster
Labor	$0.380	$0.054
Power	0.049	0.045
Repairs	0.026	0.020 (estimated)
Starting oil	0.038	.....
Coal dust	0.046	0.046
	$0.539	$0.165

A series of extraction tests for a period of five weeks gave an average of 80.3 per cent. of the silver for the old roasters and 86.4 per cent. for the new roaster. This is on the coarse crushing we have found most economical in the old type of roaster. Following is a screen sizing test on the ore fed to both roasters:

Size	Per cent. by Weight
+ 1/2 in	10
+ 1/4 in	22
+ 1/8 in	33
+ 1/14 in	15
− 1/14 in	20

The ore treated is a dense one and the recovery would be materially increased by finer crushing. With the old type of roaster this is not economical, while in the new roaster 10 mesh, and even finer product, may be handled very well.

The salt used in both cases was equal to 7.5 per cent. of the weight

of ore roasted. We are of the opinion that this can be economically reduced in the new roaster, but this has not been determined.

When compared with the roasting furnaces commonly employed for preparing ore for leaching purposes the following advantages may be pointed out.

1. There is no perceptible volatilization loss of the valuable metals.
2. There is no dust loss.
3. An ideal product for percolation is furnished even when the raw ore contains much slime.
4. Only a low percentage of an inexpensive fuel is required.
5. A high recovery without fine crushing is possible.
6. The roaster gases are concentrated, cool, and easily condensed for leaching purposes.

The Leaching Process

The leaching process is very simple. There is in fact but one mill solution, which is designated as "silver," "weak," or "barren," depending on its value in silver. The solution is made up merely of soluble salts from the roasted ore, carrying in addition about 4 lb. of free acid. The acid is in part supplied from the condensed roaster fumes, the remainder being added as sulphuric acid. At such times as the ore carries a considerable percentage of pyrite no extra acid is required. Of the salts taken up by the solution from the roasted ore, during leaching, NaCl forms the greater part. Chlorides and sulphates of the various other metals are also present.

The sluicing out of the roasters into the leaching vats is done with weak solution. This dissolves the greater part of the silver and copper from the hot roasted ore while conveying it to the leaching vat. The solution passes from the vat as the "silver" solution and most of it is pumped to the precipitation boxes. When the leaching vat is full it is leveled off, and leached with barren solution for 24 hr. It is then washed with water down to from 5° to 10° B. (specific gravity, 1.036 to 1.075) and sluiced out. The washing is regulated so as to keep the solution at the proper density. Experiments have shown this to be about 24° B. (specific gravity, 1.200). The excess barren solution is run to waste from time to time.

The best results are obtained by leaching with warm solution. During the summer months the heat from the roasted ore is sufficient to maintain a temperature of from 30° to 40° C., but in cold weather steam is blown into the solutions. This is best done in the precipitation boxes, as maximum efficiency is obtained by heating the solutions at this point.

Leaching Costs

The cost of leaching varies considerably. The minimum of 17c. was attained during the summer when the heating plant was closed down,

and the roasters were making the necessary acid. On the other hand, during the month of March when it was necessary to buy both acid and coal the leaching cost reached the high figure of 43.8c.

Precipitation

The precipitation boxes are packed with scrap iron of every description. Thin sheet scrap, old screens, etc., are desirable on account of the large surface presented. The metals in solution are deposited on the iron in accordance with their position in the electromotive series; that is, the metals of greatest potential difference are deposited first. This segregation of the metals, however, is not sufficiently clear cut to be of value in their separation.

A general cleanup of the precipitation boxes is made once each month. An intermediate cleanup is sometimes necessary on account of the boxes clogging.

The method of cleanup is rather crude, due to the present layout. The loosely adhering precipitate is brushed from the scrap iron, put into filter boxes and washed. It is then dried and sacked for shipment.

A partial analysis of the product is as follows: Au, 1.78 oz.; Ag, 6,868.5 oz.; Pb, 13.2 per cent.; Cu, 36.94 per cent.; insoluble, 7.5 per cent.; Fe, 2.5 per cent.; S, 1.6 per cent.; Zn, nil.

Our original plan was to precipitate the gold and silver on cement copper in an acid-proof press. The copper used could then be recovered on scrap iron as cement copper and returned to the precipitator. Two difficulties presented themselves. In the first place, it was hard to form and maintain a uniform cake of cement copper on the filter leaves. Secondly, the filter cloth and cake soon became impervious to the solution.

We are at present developing a method of handling the copper precipitation of the gold and silver, which promises to be both simple and efficient. However, as this is still in the experimental stage it will not be considered here.

The consumption of scrap iron amounts to about 1 lb. per pound of product recovered. The grade of scrap used costs $5 per ton delivered at the mill or 0.6c. per ton of ore treated. The total cost of cleanup, including drying and sacking, amounts to 6.5c. per ton of ore milled.

Pumps

Our practice of sluicing out the roasters makes it necessary to handle large volumes of solution. The mill solutions are corrosive. In addition to acids they carry free chlorine, and chlorides and sulphates of the various metals. No metal, commercially available, can withstand this combination, and pumps on the market, of suitable material, are either

too small or too expensive to be considered. Our sluicing alone requires a continuous supply of 140 gal. per minute, which must be elevated 89 ft.

After considerable experience with pumps that work for brief periods, and others that work not at all, we decided to install the Pohle air-lift system. These air lifts are constructed of wood pipe and a rubber air hose as shown in Fig. 3. Wood stave pipe, with bands well protected with asphalt or similar material, withstands the solutions very well.

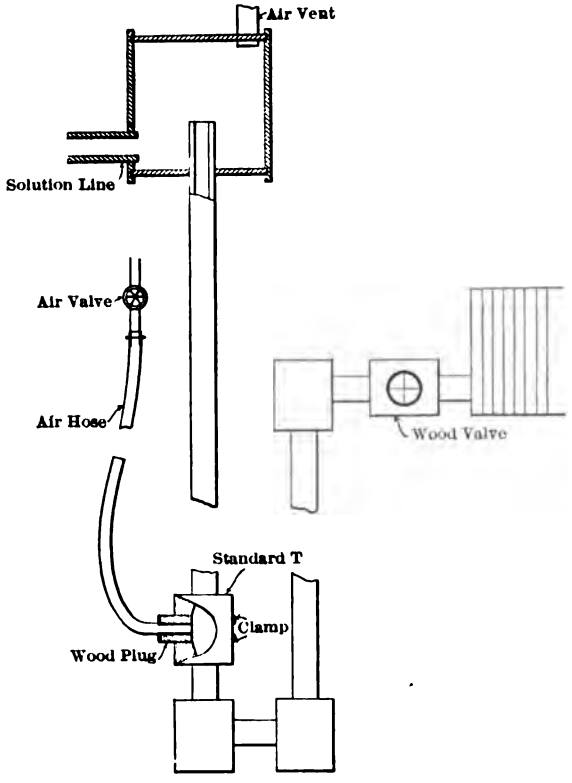


FIG. 3.—POHLE PUMP.

The highest lift of 89 ft. is made in three steps. The highest single lift is 37 ft. and this determines the air pressure used, which is about 27 lb. The air-lift pumps have been in continuous operation for over a year and have given complete satisfaction.

Every department of the mill suffers the inconvenience of being housed in an old plant not designed for the process. Furthermore, several features did not work out as anticipated, and in the consequent changes and reconstruction, convenience and economy have suffered. Our experience

has taught us many things that would add to the efficiency of a plant we might now design.

Conclusions

The roasting process is not limited to chloridizing but may be advantageously applied to oxidizing gold ores for cyanide treatment, and oxidizing and sulphatizing copper ores for subsequent leaching.

However, it is our opinion that the most favorable field of application is in chloridizing roasting. In this class of roasting we not only have the benefit of cheaper operating cost, but, in addition, have overcome the volatilization loss, which in the past has been a serious difficulty.

We have tested out samples of ore from 38 different mines. These have given us a large variety of combinations. In many cases the recovery of gold, silver, and copper has been well above 90 per cent. The ideal combination, we may suggest, is a siliceous copper-silver ore carrying 6 to 10 per cent. pyrite, and favorably located with respect to a salt supply. Under such favorable conditions, the entire cost of milling in a well-designed plant will be less than $1 per ton, on the basis of 150 tons per day. When the ore carries gold it is necessary to carry free chlorine in the leaching solutions and this adds a few cents to the cost of treatment.

Ores high in silica, iron, and manganese chloridize readily. A small percentage of lime does no harm as it is readily neutralized in the roast. A large percentage, however, is detrimental.

Discussion

F. S. Schmidt, Salt Lake City, Utah.—Any furnace that can make a chloridizing roast to yield an extraction of 92 to 93 per cent. and do this at a cost of less than 24c. per ton in a 10-ton unit opens great possibilities for the treatment of certain classes of ores. This cost does not include the salt and will be reduced to about 16½c. per ton in the 35-ton furnaces which are now being designed. On a trial run the roaster treated 9.6 tons of Ontario stope fillings crushed to pass a ¼-in. opening. The ore is well adapted to chlorination with the exception of insufficient sulphur and occasional days of high lime, but is not adapted to any other process. The following is an analysis of the crude ore: Ag, 9.52; Au, 0.029 oz.; Pb, 0.71; Cu, 0.11; SiO, 79.4; Zn, 0.57; S, 1.00; Fe, 7.9; Mn, 0.89; CaO, 1.20 per cent. The composition of the mix was:

	Per Cent.
Dry crude ore	83.4
Moisture	4.4
Salt	8.3
Coal dust	2.3
Pyrite, 70 per cent. pure	1.6

To determine the amount of volatilization, the mix was roasted in the usual way; an analysis is as follows: Au, 0.025; Ag, 9.0 oz.; Pb, 0.72; Cu, 0.13; SiO, 72.2; Zn, 0.95; S, 0.7; Fe, 7.3; Mn, 0.93; CaO, 0.73 per cent. From results obtained in an experimental roaster it has been determined that the loss in weight sustained by the mix in roasting averages about 7 per cent. The loss may be said to be the weight of the coal, moisture, and one-half the salt. The percentage of sulphur is inconsiderable. Using the factor 93 per cent. on the weight and calculating the analysis from the crude heads for comparison, we have: Au, 0.027; Ag, 8.89 oz.; Pb, 0.67; Cu, 0.11; SiO, 74.5; Zn, 0.54; Fe, 7.4; Mn, 0.84 per cent. The sulphur naturally cannot be figured. It will be seen from this that there is no measurable volatilization of the silver or gold. As a further check, if the assumption is made that the silica is constant in weight and the relation calculated between the percentage of silica and the ounces of silver in both the crude and the roasted mix, it will be found that this again shows there is no volatilization.

To determine the uniformity of roasting, nine 100-lb. samples were taken at regular intervals by cutting the discharge stream. Small samples of each of these were kept separate and a 24-hr. leaching test was made under approximate mill conditions. In all the following leaching tests the extraction can be calculated direct on the roasted mix, due to the absence of volatilization. Experiments were made to determine the loss in weight of the roasted mix in leaching and the average result is 12.5 per cent., which again can be checked by comparison of the silica percentage in the roasted mix and in the leached tails, which in this case gives 13.2 per cent. The extractions on these nine interval samples varied from 91.2 to 93.4 per cent., the tails varying from 0.6 to 0.9 oz. silver; the average roasted mix was 9.06; average tails, 0.78; and average extraction, 92.4 per cent. These results show the extraction to be high and uniform.

The next point is the coarseness of crushing for this furnace. A screen analysis of the raw mix, using 8, 14, 20, 40, and 60 mesh screens, shows that the values increase with the fineness from 6.8 to 12 oz. By weight, 34 per cent. is plus 8 mesh and 9.5 per cent. is through 60 mesh; 28.0 per cent. of the value is plus 8 mesh and 13.8 per cent. is in the through 60 mesh. A screen analysis of the roasted mix shows again that the values increase with the fineness. We now have 41.2 per cent. of the weight on the plus 8 mesh and only 5.4 per cent. in the fines, while the plus 8 mesh has 31.2 per cent. of the values and the fines 8.1 per cent. It will be noted that in the roasting the coarser sizes have increased at the expense of the fines. This is what would be expected from an inspection of the roasted product and is a very desirable result as an aid to the leaching.

A leaching test was made on these sizes separately to determine whether it would pay to do finer crushing by determining the different

percentages of extraction. The plus 8 mesh gave an extraction of 89.8 per cent. and the fines gave 96.7 per cent.; the tails in the plus 8 mesh gave 0.8 per cent., while the tails of the fines gave 0.5 per cent. It will be seen that while the fines give the best extraction, the extraction on the coarser sizes is really higher than would have been expected. This result is probably due to the nature of the furnace, with its 4 ft. thick bed of ore. It will also be seen that the question of finer crushing is solely one of the value of the ore.

To determine the relative velocity of leaching of the different sizes, a test was made by interrupting the leaching action and making a screen analysis of the imperfect tailings. For this purpose only 3 parts by weight of mill solution to 1 of ore were used instead of the usual 6 of solution to 1 of ore. The coarser products gave tailings running from 1.2 to 1.8 oz., while the fines retained a value of 3.0 oz. The percentage of extraction on the plus 8 mesh was 79.7, and on the fines 80.7. These partial tailings were then leached again to the full extent and the extraction on the plus 8 mesh increased to 87.3 and on the fines to 95.5 per cent. The tailings decreased regularly from 1.0 oz. in the plus 8 mesh to 0.7 oz. in the fines. The second leaching reduced the value of all sizes, but mainly the fines, which resist the leaching the most. The question of fine grinding is therefore important. From tests made on numerous ores it is safe to say that this type of furnace will give more effective chloridizing on coarser sizes than furnaces using a thin bed of ore.

Another point in connection with this roaster is the ease with which it is possible to get an excellent physical constitution of roasted ore for the subsequent leaching. The gathering up of the fines in roasting and their release only as the values are leached out is well shown by a study of the screen test of the tailings. The percentage of value of the fines is 16.1, while if the screen analysis is made after only half the solutions have passed through we have 7.9 per cent. The percentage of weight is the controlling factor, because the tailings run very uniform for the different sizes. In the crude ore the weight of the fines is 9.5 per cent.; in the roasted mix this is reduced to 5.4, showing the effect of gathering the fines. In the partly leached ore some of the fines have been freed, to the extent of 9.7 per cent., while in the thoroughly leached mix the amount has been increased to 15.5 per cent. An analysis of the tailings gave the following: Au, 0.005; Ag, 0.75 oz.; Pb, 0.41; Cu, 0.06; SiO, 83.2; Zn, 0.8; S, 0.2; Fe, 4.5; Mn, 0.62; CaO, 0.52 per cent. It will be seen from the low tenor of copper in this ore that it is very difficult to get sufficiently accurate determinations of the copper for close figures on extraction. But extractions of over 90 per cent. have been obtained on ores of better defined values. The mill solutions contain 8 lb. metallic lead per ton of solution, the profitable extraction of which has not yet been worked out. The lead is in the form of chloride.

It would seem from my results and an analysis of the cost data that this furnace is a cheap and efficient chloridizer; that it is well adapted to the treatment of gold-silver-copper ores having a siliceous gangue, and opens a field of great possibilities.

OLIVER C. RALSTON,* Salt Lake City, Utah.—Mr. Holt has given some costs for roasting (16.5c. per ton) which are so surprisingly low that they call for comment. Mr. Holt has informed me that this cost does not include the cost of salt, which is a variable factor rather hard to approximate. This, taken in conjunction with the low price paid for coal dust, makes his low figure more easily comprehended. However, it might be well to call attention to the fact that although this coal dust is a waste product, loaded on to cars for the cost of the labor necessary to do it, if chloridizing leaching reaches a very extensive application the coal men will be liable to feel that they are entitled to a little more than 25c. per ton for this product. Still, if the Holt-Dern roaster can roast ore at 25c. per ton, allowing for higher cost of coal dust, there will be nothing to complain of, and Messrs. Holt and Dern are to be congratulated on their solution of the mechanical difficulties in applying this kind of a roast.

Another point about this roaster is that it seems to be designed along the line of good counter-current principles so as to act as a heat exchanger. The fuel is burned in the ore, as in the blast furnace, affording a maximum of heat absorption by the ore. We find the air blast being preheated by the hot roasted ore before it reaches the zone of combustion and then the hot gases of combustion in their turn preheat the ore before it reaches the combustion zone. And, moreover, the machine is simple and inexpensive. It would look as though this roaster should have a future.

Outside of the new roaster, the process will be found to be nothing but a new modification of the old Augustine process, used in days past in the hydrometallurgy of silver. Nothing but the roaster is subject to patent and the rest of the process is old enough to be free to all. In Utah there is a great deal of ore that is amenable to this treatment and doubtless more mills will be built now that we are getting some assurance that the process pays. Likewise there is plenty of such ore elsewhere in the United States. It is an interesting sight to see a process as old as the Augustine, which had seemingly been hurled into Aphelion by the introduction of cyaniding, now coming back to claim its own.

Two drawbacks, however, are to be met with, and one of them Mr. Holt has already mentioned; namely, the deleterious effect of lime. There is plenty of ore within a radius of 50 miles of where we sit which will be found to contain a fatal amount of lime. The other drawback is that, as of old, we cannot recover the zinc of the ore. Mr. Holt informs me that he does not know whether the zinc of the ore is chloridized and goes into

* Non-member.

solution or not, but it ought to. At one of the plants of the Pennsylvania Salt Co., where they have roasted pyrite cinder for 25 years, the zinc content is definitely known to be chloridized and accumulates in the leach solution, and for years was finally discarded into the sewer at the rate of several thousand pounds of zinc per day. Recently efforts have been made to recover this zinc. The consumption of the salt by this zinc and its fouling of the solutions and ultimate loss, while not serious at the Ontario, is bound to become a serious problem with the further application of the process.

There are a number of questions which I would like to put to Mr. Holt and they are as follows:

It is stated in the paper that ores amenable to this type of treatment are those with high percentages of silica, iron, or manganese. Is there a lower limit of silica allowable? Must the iron be as sulphide or oxide? What is the advantage of the presence of pyrite, is it necessary, and if so in what percentages? How can high manganese cause the ore to chloridize well?

What is the maximum allowable amount of lime?

How can one determine the amounts of salt and of fuel necessary for this roast?

The paper states that the driers reduce the moisture to 5 per cent., but that there is considerable variation from this amount. How much is the variation? In other words, how much is the allowable variation in moisture content of the ore on roasting?

How many tons of solution and of wash water are used in the Ontario mill to one of ore?

Does the Holt-Dern roaster handle a finely crushed ore?

THEODORE P. HOLT, Park City, Utah (communication to the Secretary*).—In reply to Mr. Ralston's discussion, I must confess my inability to answer all his questions. Some of them would require generalizations too broad for our present limited experience.

Silica, iron, and manganese are indicated as desirable gangue minerals in an ore to be chloridized. Silica at proper roasting temperatures is for the most part inert. Iron and manganese are generally recognized as active chloridizing agents. The iron is desirable either as oxide or sulphide, preferably as sulphide. Pyrite is very desirable though not always necessary. In the first place, it is an active chloridizer of silver and copper. Secondly, it converts $CaCO_3$ and CaO into inert $CaSO_4$. In the third place, it is valuable as a fuel and when present to the amount of 6 per cent. or over no additional fuel is required.

The deleterious effect of lime in chloridizing metals is well recognized. When the chloridizing roast is followed by an acid-salt leach, as in

* Received Sept. 21, 1914.

our case, it is doubly essential that all $CaCO_3$ and CaO be converted into neutral compounds; otherwise the consumption of acid would be prohibitive. We have secured satisfactory results on an ore carrying 23 per cent. CaO by analysis. But since the ordinary analysis reports the calcium combined as sulphate and silicate as well as the carbonate, it is of limited value in determination of probable results.

The "amounts of salt and fuel necessary for the roast" can be determined by experiment on a small sample. The amount of both reagents in the mill will be slightly less than in the small laboratory roaster.

The moisture in the mixed ore, while essential, may be subject to considerable variation without serious effects. With the new roaster the variation may be from 4 to 9 per cent. in many cases without seriously affecting the results.

Mr. Ralston asks if the Holt-Dern roaster can handle finely crushed ore. This depends on the characteristics of the individual ore, and also on the size he designates as "finely crushed." We have found some ores to work well when crushed to 20 and even 30 mesh. However, we must bear in mind that one characteristic of this roaster is its ability to chloridize well coarsely crushed ore, thus avoiding the expense of fine grinding. This has been demonstrated in a great many ways. I will mention one by way of illustration. A large sample of ore crushed to pass ¼-in. screen was cut in two samples. Sample No. 1 was passed through the Holt-Dern roaster carrying a 4-ft. column. Sample No. 2 was roasted in a bed only 12 in. deep. Leached under the same conditions, sample No. 1 gave 92 per cent. extraction, while sample No. 2 gave only 83 per cent. extraction. The difference is due to the fact that sample No. 1 was in intimate contact with the chloridizing gases at a chloridizing temperature for over 4 hr., while for sample No. 2 these conditions lasted less than 1 hr. The prolonged heating is effective in opening up the coarse particles of ore, thus exposing any inclosed mineral particles.

Since preparing the paper on Chloridizing Leaching at Park City a successful silver precipitator has been put in operation at the plant. This makes possible the marketing of a high-grade bullion in place of the base silver precipitate, thus materially reducing the marketing expense. Before entering the iron boxes as shown in the flow sheet, the pregnant solution is first pumped to two cement tables, each 8 ft. wide by 40 ft. long. These tables have a slope of 1 in. to the foot and carry a thin layer of scrap copper, upon which the silver precipitates. Practically all the silver remains on the table until clean-up time. At the lower end of the tables is a settling tank. Twice each month the silver product is sluiced into this tank and the excess water decanted. It then passes to a filter tank, where it receives a final wash. The dried product carries 60 to 70 per cent. silver and melts easily to bullion 970 fine.

"Playa" Panning on the Cauca River

BY WILLIAM F. WARD, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

ONE often reads of the rich placer gravels in many of the canoe-traveled rivers of South America. The apparent richness of these gold-bearing gravels impresses the traveler, and in fact he may see batea after batea showing enough colors to figure over a dollar per cubic yard.

These workings are called "playas" from the Spanish word *playa*, meaning shore or beach. Many workings have been reported in tropical rivers, but to my knowledge no dredge has made a success in such deposits. Investigation generally shows that the area of richness is limited and especially that the depth of the rich concentration is hardly ever more than a few inches.

The Cauca river in Colombia, South America, is noted for its many rich playas. This river rises near Popayan in the southern part of Colombia, flows north through a narrow valley between parallel mountain ranges and empties into the Magdalena river near Maganque. The air-line distance between source and mouth of this river is only about 500 miles. The rainfall is large and the lower Cauca appears to be about the size of the Missouri river in the United States. From its mouth up to Valdivia, in the State of Antioqua, it is navigable. Above Valdivia is a narrow gorge, above which the river is again navigable for some distance.

The part of the Cauca where the playa panning is most noticeable is from Valdivia down to where the Nechi river enters the Cauca, a distance along the river of about 100 miles. The current in this part is swift, so that the stern-wheel river steamers make the trip with some difficulty.

In the course of my work in this section of Colombia I have had the opportunity to make a study of several of these playa workings and will give in detail the conditions observed.

The Island "Playa"

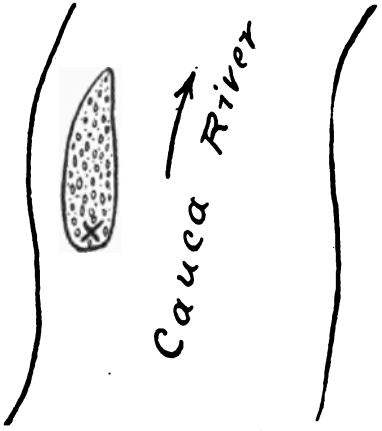
The richest of the island playas is about 40 miles below Valdivia and a little below the old town of Caceres. The shape of the river and the location of the deposit are shown in Fig. 1, which indicates the condition at low water. The river at this place is about 600 ft. wide and the island about 75 ft. wide by 300 ft. long. The island is made up of coarse pebbles, some of which are as big as a man's head.

In times of high water this island is under several feet of water and in a swift current. When the river lowers the current is still fast enough to prevent sand or silt from covering the coarse pebbles.

At high water the banks of the river for several hundred feet above the island are continually caving in, and some of this material, together with the usual drag of fine gravel along the bottom of the stream, passes over this submerged island.

The coarse pebbles forming the surface act as natural riffles. In the course of a few months a mass of material is thus handled by natural agents, and a rich concentration of fine gold results. The natives work the playa when the water is low enough, which is two or three times a year.

As soon as the water goes down enough to permit wading on the island, the natives turn out with their small hand-shaped picks and their wooden bateas. They scrape the fine gravel into the bateas, rough out the coarser pebbles, then wade to the shore and complete the panning in the quieter water.



Island surface made up of coarse pebbles. × Richest point of gold concentration.
FIG. 1.—SKETCH OF CAUCA RIVER BELOW CACERES, SHOWING "PLAYA" AT LOW WATER.

Many natives work at this place, so that by the time the low-water season is over the whole surface of the island has been gone over and only the coarser gravel remains, thus leaving the natural riffles thoroughly cleaned. At times it happens that the playa so worked over is even richer the following season.

At this playa a native for the first two or three days makes as much as $3 per day. The work continues for several weeks until only about 30c. a day per native is realized, when operations are abandoned.

The Typical Beach "Playa"

Another noted playa just above Caceres is worked by the same natives; but it does not yield as much gold as the island playa. This is more nearly the typical playa or beach deposit, and is shown in Fig. 2.

The west side of the river is a beach of coarse pebbles which act as natural riffles. The river above turns in a way favorable to sending the

drag of small gravel over the coarse beach gravel. The current at high water is swift enough to prevent sand or silt from accumulating, but is not so swift as to prevent some of the fine gold from depositing. Most of the playa deposits on the river are of this type, and are workable to a certain extent at all times except at extreme high water.

Cauca River Conditions Favor Playa Formation

The Cauca river is large and flows through a long narrow wooded valley, which makes it practically impossible for a heavy rainfall to

FIG. 2.—A TYPICAL BEACH "PLAYA" ON THE CAUCA RIVER NEAR CACERES.

produce a damaging flood. The greatest difference between high and low water is 10 ft. and usually it is much less. Had the valley been basin shaped resulting conditions would be quite different. The locations of many river playas remain the same year after year and only change when a marked change in the main currents takes place.

Near the mouth of the Tamana creek, about half way between Valdivia and the mouth of the Nechi river, a small arm of the river, which had been closed for many years, was opened by extra high water; as the entrance contained coarser pebbles than usual, a very rich concentration took place. Here for several days as much as $4 per day was obtained by each native worker.

Once in a while a large snag will cause a change in the main current and incidentally a local playa will be formed below.

In the upper and swifter part of the river near Valdivia the shore line at low water shows boulders and angular rocks. These places are worked periodically by the natives, a small amount of gold being obtained.

The extreme lower parts of the river show playas only at very low water, the coarse gravel only showing at such times. One of these playas about 5 miles above the mouth of the Nechi river compared favorably with the playas in the upper river, since the water had not been low enough to permit working for several years.

The right to wash gravel on the playas is free. So when the high-water season is over the natives go prospecting in canoes for the richer playas. When a good one is found the news spreads and soon the richest places are cleaned up.

Most of the gold is obtained from the playas by women workers. While the pannings are rich many native men are seen on the playas; but the women keep at it longer, for they are content with less renumeration for their labor.

The bulk of the gold sooner or later gets to the Caceres storekeepers, the smaller merchants along the river, who make trips to Caceres once every two or three months, buying the gold at the various playa camps on the way. An average of about 15 lb. of gold is bought at Caceres monthly, about two-thirds of it coming from the river playas. Roughly about $35,000 is, therefore, obtained yearly from the Cauca river playas.

From several drill holes in the river gravel itself near Caceres it was found that the value per cubic yard was from 1c. to 3c. Even this small amount of gold in the river gravel itself is enough to account for the source of the gold in the playas. This gold is small in size, often a dozen "colors" to the milligram; but, as is usually the case with such gold, it has a fineness of over 900.

One drill hole in the lower part of the river on an island near a rich playa gave results as follows: About 5 ft. of barren loam and sand; then about a foot of good gravel of 30c. grade; then to a depth of 50 ft. material of poor grade. The total hole averaged only about 2.5c. per cubic yard. The hole was not completed to bed rock.

The bench gravels on the sides of the river seem to have coarser gold and at places there are some very rich channels. At a playa just below a place where bench gravel had been caving into the stream some coarse gold of a dull color was seen in the pannings along with the fine bright river gold.

There are evidences of extensive work in the bench gravels all along this stretch of river. A fair record of gold produced is credited to the district. Not enough prospecting has been done so far to determine whether or not the areas of pay gravel are large enough to work by modern methods.

The rich channels of the benches will be first worked out, but the playas for years to come will be a source of gold.

The Descriptive Technology of Gold and Silver Metallurgy

BY A. W. ALLEN, LONDON, ENGLAND

(Salt Lake Meeting, August, 1914)

THE technological study of the treatment of gold and silver ores has been largely responsible for the phenomenal strides which have marked the progress in this branch of metallurgy during recent years. In no other application of science to industry is system more imperative at every stage; correct formulation of result is only second in importance to efficiency of operation.

Progress in method, on the other hand, is mainly due to individual initiative, aided by the work of technical journals and metallurgical societies.

There still remains a field for profitable discussion, and it is my intention to deal with some inconsistencies of expression common to a number of writers on the subject; and also to draw attention to the false impressions created by a statement which may, possibly, be only unintentionally misleading. I also take the opportunity of tracing the source of both metallurgical and economic result; and would question the desirability of the methods usually employed in arriving at a final statement.

In the first place, I wish to deal with the question of extraction. This term may mean anything or nothing, and it is often used to mislead. In a report of a mining engineer as to the result of operations subsequent to the remodeling of a treatment plant prominence was given to the statement that the extraction had amounted to 91 per cent. On examination it was found that only a small percentage of the so-called extraction was calculated on actual bullion return, a proportion being figured from the gold left in the zinc boxes, and not recovered. In addition to this, a proportion of the extraction was said to be due to gold in concentrate. As there was no concentrating apparatus on the property and as concentration did not form a part of the scheme of treatment the matter was investigated, with the result that the concentrate, of lower gold content that the original ore, was found to be the result of an inefficient pumping system which failed to provide for the whole of the battery product delivered to it. The balance was stacked and, although itself of no marketable value, the gold content was allowed to form a basis of extraction figures.

The illustration may be extreme, but it will serve to show to what lengths an engineer may go in the mutilation of the most vital term in metallurgical technology.

An ore may be treated experimentally with cyanide solution and 90 per cent. of the gold or silver dissolved. This may be quoted as an extraction of 90 per cent., but it is preferable not to do so, since it is, at best, only an experimental extraction; and by no parallel method of operation in practice could such an "extraction" be actually realized as a recovery. The percentage obviously refers to solution, and the result of the test might as well be stated as such. In practice, an assay or theoretical extraction may be a valuable indication as to the progress of operations, but it is of no value for the purpose of economic comparison unless it happens to check exactly with actual recovery. Again, an ore may be treated by a system of concentration and 70 per cent. of the gold or silver "extracted." Such an extraction is still further removed from actual recovery of metal or its equivalent money value. When the actual weight of the concentrate and the assay of the same are available for computation then the result may be termed an extraction; but when the amount of metal associated with the mineral is determined by a difference in assay content of the ore before and after concentration then the result is obviously only an assay extraction.

In the case where the final product from the reduction works is solely in the form of bullion the statement of result is, comparatively, an easy matter. The assay extraction shows the amount of metal which has been removed during treatment, as indicated by the difference in assay results. The actual recovery must necessarily be based on the actual amount of metal recovered in the form of bullion. In other instances where a proportion of income is derived from the direct sale of bullion, and a proportion from the sale of a concentrated product, then the assay or theoretical extraction is obtained, as before, by estimation from the difference in assay result before and after concentration and cyanidation. As a check upon this result the combined valuation of the bullion produced, together with the gold and silver indicated in concentrate returns, may be calculated as an actual extraction. Under such conditions actual recovery figures are out of the question; and any statement which adds bullion valuation to smelter returns of concentrate treatment should contain a definite designation of the total as an extraction and not a recovery. Extraction and recovery are terms which are not necessarily interchangeable and the time is ripe for due recognition of the fact.

The final result of metallurgical operation should be stated in a form bearing a definite significance and capable of general application for comparative purposes. By common consent it is invariably reported as a percentage of the original metal content in the ore. At this stage of the discussion it will be necessary to introduce other considerations which will be dealt with in detail. In the statement of assay or theoretical extraction two figures alone are necessary: namely, the result of the assay of the feed and of the residue. The latter figure is generally easily

obtained and, for reasons which will be dealt with later, can usually be relied upon. With regard to original metal content in the ore it may be noted that, with most installations, a reliable sample can be obtained by automatic methods after milling. Where the ore is being milled in a cyanide solution containing gold or silver the situation is complicated. It is imperative that the personal element should be eliminated in the sampling of any unground feed and, to this end, automatic sampling and splitting of a considerable proportion of the tonnage handled are imperative. With a reliable knowledge of the metal content of both feed and residue the control of operations is considerably simplified; and the figures obtained can be subsequently checked against the actual extraction or actual recovery figures obtained by direct valuation of bullion, concentrate, or other final product; or from smelter returns giving actual metal content in such product.

Theoretical extraction may be calculated daily, but in order to average the figures obtained a knowledge of the tonnage being handled is necessary. With these data a forecast can be made as to probable yield.

In dealing with actual extraction or actual recovery it has already been pointed out that the total amount of metal ultimately produced can be found by direct valuation. Two other figures are needed before the percentage of actual extraction or actual recovery can be estimated. One of these refers to tonnage, of which a daily estimation is imperative. The difficulty of calculating the actual weight of dry ore going to the mill has already been discussed elsewhere[1] and it is generally conceded that tonnage calculations are best made after the ore has been ground and by means of direct calculation or specific gravity methods. A disadvantage in the latter method of tonnage estimation has been suggested by T. T. Read,[2] who refers to the possible formation of colloid hydrates during fine grinding. I should like to see further discussion on this point and in the meantime I would tentatively suggest that the emulsoid state may not be induced by simple comminution. Recent research would seem to allow us to assume that all substances occur in the colloidal, if not in the crystalloidal state; and from Cornu we learn that "the gels of the mineral kingdom are the typical products of every normal weathering process." I do not think that any amount of fine grinding could transform kaolin into emulsoid clay.

In connection with the specific gravity method of arriving at tonnage figures it may be added that, in the case where slime treatment is continuous, there are obvious advantages to be gained by the inclusion in the system of two tonnage tanks, which will incidently act as agitators,

[1] A. W. Allen: The Estimation of Tonnage, *Mining and Scientific Press*, vol. civ, No. 6, p. 308 (Feb. 24, 1912).

[2] The Estimation of Tonnage, *Mining and Scientific Press*, vol. civ, No. 12, p. 443 (Mar. 23, 1912).

and which can alternately be filled with and emptied of pulp. From these the actual dry tonnage being handled can be obtained by specific gravity methods.

In addition to the tonnage figure, an additional estimate, dealing with gold and silver content either before or after treatment, will be needed before actual recovery percentage can be calculated. Such a figure must be reasonably accurate and for a number of reasons the assay of residue should be used for this purpose. This particular sample can be relied upon as being more representative than the feed sample for the following reasons:

1. The ore is finely ground and thoroughly mixed during treatment before being sampled.

2. Coarse metal of value has been removed, thus reducing the possibility of error in assaying.

3. The final treatment process allows of a correct average sample being taken from a large bulk of material by simple means.

4. Where classification is efficient there is no chance of the personal element entering into the question; neither is it possible to vary the metal content at will except in the rare instance where residues are discharged with different percentages of moisture, the latter being in the form of valuable solution.

An additional reason why reliance may be placed on the residue sample lies in the fact that the figures obtained from the assay of such material may, in the great majority of instances, be regularly and periodically checked by fresh sampling and assaying. This is an important point.

A further advantage is seen in instances where, if the assay of feed had been used, an actual extraction at times of over 100 per cent. would be recorded. There is always a possibility of the actual extraction reaching or exceeding the theoretical or assay extraction, but there is no possibility of the actual extraction reaching or exceeding 100 per cent. A logical method of arriving at final result should preclude the possibility of absurdity in statement.

The most important phase of descriptive technology dealing with metallurgical operations undoubtedly refers to final result; and in this connection the value of clarity of statement is obvious. I recently attempted to systematize the information, published in book form, and dealing with metallurgical operations on a number of important properties. I found that no comparison of result was possible. In one instance only was a serious attempt made to give information to the reader. In this case the percentage of gold recovered as bullion was calculated from the average assay content of the original ore multiplied by the tonnage treated. In three instances the "extraction" referred to certain assay results, and was obviously only an assay extraction. In two instances there was no indication as to how the published extraction figures had

been obtained. In two instances the results were either contradictory or were based on figures bearing no connection whatever with the actual metallurgical extraction. In the final instance the phrase "metallurgical recovery" had been used to include the metal in the rich ore sorted at the breaker plant, the metal shipped in the concentrate, and the metal really recovered as bullion. In this instance the word "recovery" seems to have been more sadly misused than the previous term "extraction." The percentage figures tell us nothing as to the metallurgical efficiency of the plant or the suitability of the scheme of treatment. As the amount of ore shipped to the smelter rises so the "recovery" increases. By shipping all the ore to the smelter the "recovery" would reach 100 per cent.

A definite standardization of such results is badly needed, and accounts of metallurgical operations would have an added interest and value if accompanied by a clear and concise *résumé* of the actual metallurgical result of such operations. Actual recovery might well be stated in all instances where the final product is in the form of bullion. Actual extraction might be stated in all cases where the final product is in a less concentrated form, and where the yield consists of bullion as well as concentrate, etc. If extraction and recovery figures are unobtainable then a plain statement of the fact is alone necessary. Assay extraction by another name is of little or no value except when it coincides with actual recovery. In the latter event it might just as well be stated as an actual recovery. Assay extraction can always be checked and it is the actual extraction or actual recovery which alone is of economic significance and metallurgical importance.

On the question of the statement of metallurgical result I venture on some definitions:

1. *The assay or theoretical extraction* is the amount of metal which is indicated by assay results as having been removed from the ore during treatment.

2. *The actual extraction* indicates the amount of metal removed from the ore and isolated either as bullion or in other concentrated form. Such actual extraction is estimated by the direct valuation of the bullion and the statement of return from smelter or buyer showing actual metal content in concentrated product, upon which the sale was effected.

3. *The actual recovery* can only be based on the direct valuation of bullion, and the term is only available for use when the bullion produced is approximately the same market value, *per se*, as the contained gold and silver.

In the calculation of percentage extraction or recovery the following summary will explain the method of statement:

Assay Extraction

Where a = the assay of feed,
z = the assay of residue, and
t = the tonnage represented,

then $\frac{100\ (a-z)}{a}$ = *percentage assay extraction,*

and $\frac{a_1t_1 + a_2t_2 + a_3t_3 + \ldots}{t_1 + t_2 + t_3 + \ldots}$ = average feed assay = A,

and $\frac{z_1t_1 + z_2t_2 + z_3t_3 + \ldots}{t_1 + t_2 + t_3 + \ldots}$ = average residue assay = Z,

then $\frac{100\ (A - Z)}{A}$ = *average assay extraction percentage.*

In the case where an appreciable tonnage is associated with the metal extracted the average assay content in residue *per ton milled* must be calculated and used in the formula, otherwise actual extraction will exceed assay extraction.

Actual Extraction

Where M = the amount of metal in bullion, concentrate, or precipitate,
T = the tonnage milled,
t = the tonnage of concentrated product removed, and
Z = the average actual residue assay,

then $\frac{100\ M}{M + Z\ (T - t)}$ = *average actual extraction percentage.*

Actual Recovery

Where M = the amount of metal in bullion,
T = the total tonnage, and
Z = the average residue assay,

then $\frac{100\ M}{M + TZ}$ = *average actual recovery percentage.*

It is, of course, assumed that the same unit of weight would be used throughout in the calculation of metal content in ore, bullion, concentrate, precipitate, etc.

Any metal in a concentrated product removed from the ore after or during reduction, and the value realized elsewhere by direct sale or otherwise, may be included in the statement of actual extraction. The total amount of metal so extracted would form a basis of calculation for percentage actual extraction. On the other hand, the metal in rich ore abstracted before treatment of the main bulk can form no part of, nor may it be used to influence, a synopsis of such treatment. Its cost of extraction cannot be included as an expenditure *per ton milled* and its content cannot be added to any statement of recovery made in the reduc-

tion plant without falsifying figures which might be obtained dealing with the result of actual metallurgical treatment.

The next consideration refers to the case of calculation of extraction from ore being treated for both gold and silver content, and in the case where both metals occur in quantities of economic significance. A numerical average is of no significance, neither is extraction based on combined metal content before and after treatment of other than metallurgical import. The extraction in current metal value is sometimes of use for comparison, but such a method of statement is not of value for general purposes. The reason for this lies in the fact that the result is influenced by market fluctuations which are beyond the control of the metallurgist The only exception to this occurs in the rare instance when the percentage extraction of both metals is identical. The extraction and recovery percentages should be stated in such a manner that an alteration in the price of one of the metals would not necessitate an explanation as to a rise or fall in the extraction or recovery figures caused solely by such fluctuation. Extractions of gold and silver are, therefore, best stated separately.

With regard to the system to be used in the statement of gold and silver content in ore and bullion, it must be admitted that the metric system is preferable to any other. In troy measurement the units bear no definite relation to one another, and there is no regular factor of difference. The use of the troy grain is now practically discontinued, and the juxtaposition of the facts that 24 grains equal 1 pennyweight and 20 pennyweights equal 1 ounce has often led to serious errors in calculation. It is perhaps too much to hope that the centimeter-gram system may be extended to all calculations used in ore treatment within the lifetime of the present generation.

In most English-speaking countries there seems no objection to the use of the troy ounce in the calculation of silver content in ores, and it is in general use in connection with bullion in most places. The logical equivalent for the use of gold ores is the pennyweight (dwt.), which is perhaps preferable to the use of the decimal subdivision of the ounce. The use of the American dollar as a unit of gold content is illogical, although in the technical literature of the United States its use is practically compulsory. It is common to see a statement that a gold ore has a value of $1. This may mean that the ore assays 1 dwt. of gold, or thereabouts, and this metal if completely recovered in its original state of purity would have a value of $1. The original statement is really a jumble of contradictions, because the ore may not be worth 10c. when placed on the market. The statement of metal content in terms of current coinage is a difficult matter, and verbosity is unavoidable if logic and precision are not to be entirely disregarded.

Experimental and Metallurgical Report Work

The remarks previously made with regard to extraction are equally applicable in connection with experimental work. Extractions of metal based on differences of assay results should be definitely stated as assay extractions. When they refer to differences in gold and silver content as a result of experimental cyanide treatment such extractions are preferably referred to as solution extractions. They should, in all cases, be distinctly differentiated from probable or possible recovery. These latter figures are only obtainable in the case where the experimental plant is capable of duplicating actual practice in every detail, and where the ore being assayed is not freed from the normal amount of dissolved gold or silver generally found associated with it after ordinary treatment. When preliminary work is carried out on a small scale and where there is no working size filtration or leaching plant available, the assays of the washed ore must be used for the purpose of computation of possible recovery figures. The following ratios are available as a rough guide for this purpose:

Experimental Assay Extraction, Per cent.	Scheme of Treatment	Probable Recovery, Per cent.
100	Amalgamation	100
100	Leaching of sand with precipitated solution	99 to 99.5
100	Sliming, followed by continuous replacement with barren solution during agitation. Filtration of pulp	95 to 98
100	Sliming, followed by filtration and washing	90 to 98

These figures introduce the consideration that the more complicated the treatment and the finer the ore is ground the lower becomes the actual recovery of metal. This point is invariably overlooked in reports of experimental work where solution percentage is confused with possible recovery.

The Statement of Working Costs

In conclusion, I would refer briefly to the question of the statement of working costs, and in this connection I would urge the importance of the local segregation of working cost from general expense, in order that operators in positions of responsibility should be given every opportunity of keeping those expenditures over which they have control under constant scrutiny, so that continued economies in total working costs may be effected and clearly indicated in the returns.

On the question of amortization, or the liquidation of the original cost of the plant by means of payments out of profits of treatment, there is little to be said, because in general mining work the provision of such a sinking fund has been deemed unfeasible. There exists, however, no logical reason why theoretical amortization figures should not be taken into consideration in the first place when some definite scheme of treat-

ment has to be decided upon. In the latter event two factors alone are generally supposed to influence the decision: the probable recovery, and the cost of treatment. If amortization costs were added to the cost of treatment in each case and the matter considered from a strictly business standpoint, it is obvious that these charges, coupled with the necessarily higher operating costs, would afford a sound reason against the selection of any process which involved an unnecessarily extensive equipment, involving the necessity for comminution of the ore beyond an economic limit. Even if amortization charges are not feasible in actual practice, there is no reason why they should not be considered in preliminary estimation work.

On the question of repairs and renewals, I would suggest that in a statement of working costs these two items be distinctly differentiated. I would even go further and suggest that items coming under repairs should be added after the total working costs have been arrived at. My reason for this lies in the fact that actual operating costs are for the purpose of locally checking extravagance and unnecessary expenditure and for the encouragement of the introduction of economies. I would point out that it seldom occurs that mill superintendents who are held responsible for working costs are also responsible for either the scheme of treatment or the initial plant arrangement. Repair costs are largely an index of the class of plant erected in the first instance; and if such an item were isolated and added to the total working costs it would form a powerful argument against misplaced economy in the first instance.

In the statement of construction costs I would urge the necessity for the exclusion of all sums paid for wearing parts from the final statement of cost. These should be segregated out, placed to a suspense account, and charged out as operating expense. The absurdity of charging everything required to complete the plant to construction has been exemplified in every instance, but the practice is still in general vogue. In one instance which came under my notice a ball-mill plant ran with an entire avoidance of liner expense, the principal running expenditure, for over five months. In another case an extensive conveyor plant ran for over two years with no belting expenditure charged out to working costs. As soon as the original equipment deteriorates there is a rise in operating cost, due to the necessity for replacement. Such fluctuations are wholly inadmissable and convey false impressions, and I have always found that the last thing to be blamed is the system of book-keeping responsible for such anomalies.

In placing before the members of the Institute the foregoing remarks on the descriptive technology of gold and silver ore treatment I realize that the subject has been dealt with from a single viewpoint only—that of the practical metallurgist. Many of my contentions are admittedly argumentative and I trust that a critical discussion will tend to throw more light on the subject.

The Dorr Hydrometallurgical Apparatus

BY JOHN VAN N. DORR, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

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Introduction

It is 10 years this summer since the first of the contributions which it has been my privilege to make to the working tools of the hydrometallurgist was set at work, but a full description of what has come to be known as "Dorr Machinery" has never been published in the *Transactions* of the Institute, and it has been suggested to me that an account of the same would prove of interest to members, although most of those engaged in cyaniding and concentration are probably familiar with it. The account which I shall give of the development of the machines must be recognized as being written from the viewpoint mainly of my own local experience and not from that of one who was familiar with the general metallurgical practice throughout the world. Most advances in any art are conceived before the time is ripe for their development and any invention which comes into general use will usually be found to have been tried out in a crude way and abandoned more than once. If the early experimenter went as far as the Patent Office, his work may serve no other purpose than to prevent the issuance of the broadest patent to those who later may develop a commercial process.

The cyanide process in the United States early in 1904 was mainly confined to the leaching of dry-crushed ores, both coarse and fine, and sand treatment of tailings from concentration and amalgamation. In the Black hills some plants were crushing in cyanide solution, leaching sands and handling the slime by the well-known decantation process.

Filter pressing, in use for some years in Australia, had been tried in several places in this country and the Moore process had been abandoned at Mercur after a short run, but was doing good work at the mill of Lundberg, Dorr & Wilson, at Terry, and being installed at Bodie and the Liberty Bell.

A general feeling, however, still existed that slimes were to be avoided when possible, on account of the losses in treatment and the difficulty of making a good leaching product when they were present in large amounts, especially when the crushing was done in cyanide solution.

The Dorr Classifier

History

The object of the first classification done in cyaniding work was to produce a leachable sand and overflow as small an amount of sand as possible with the slime, which was usually run to waste.

In the early days various arrangements of adjustable tank gates were used, and sometimes direct overflow of tanks filled with a rotary distributor sufficed. Later, cone classification, of which the Merrill system, as

introduced at the Homestake, was probably the most perfect, was adopted. Crushing in cyanide solution, however, first used successfully in the United States by J. M. Henton about 1901, added materially to the difficulty of cone work, especially in small plants on soft and variable ores, and in the Black hills at least caused a search for other methods.

An early attempt to solve the problem that was considered very successful at the time consisted of a box about 3 ft. cube, situated over each leaching tank and equipped with a hinged bottom. The stream of pulp was fed into this box and the solution and slime overflowed until the box was full, when the mill man released the bottom and the sand fell with a splash into the vat below, while another charge was collected.

I am told that this method was used in Rhodesia years ago and brought good profit to the inventor. It might be regarded as a poor apology for the double-treatment tanks in use in South Africa and in the United States.

A description of a small plant in the South Atlantic States, which appeared in one of the mining weeklies in 1905, shows that the problem was not considered satisfactorily solved at that date. The writer somewhat proudly told how he had overcome the difficulty of securing clean sand by flowing his pulp into mortar boxes alternately and shoveling the settled sand therefrom into his leaching vats.

These boxes were later abandoned for double cone classification of the sand, which worked well on quartzite ore but gave great trouble on soft material, especially in small plants with irregular feed.

The cone system was installed in the 100-ton Lundberg, Dorr & Wilson mill above mentioned, but on our ore, which varied from a soft, clayey shale to hard quartzite, it was impossible even with a man constantly watching the cones to get a good leachable sand and avoid overflowing sand with the slime. This resulted in heavy losses through bad leaching of the sands and great difficulties in operating the Moore process on the slime.

I had felt for some time that a mechanical method of separating sand, that would not require constant attention, was feasible, and realizing that the success of the property, which was operating on a close margin, depended on good classification, proceeded to attack the problem. I knew of the work of George Moore in washing slime from the roasted fine ores at Mercur with a spiral lined cylinder and later with a screw conveyor in an inclined trough, but both methods were somewhat discredited by the abandonment of the Moore process itself at the Mercur plant.

A description by J. E. Johnson, Jr., in the *Transactions* of the Institute, of a sand shovel he was using to remove sand from a tailings launder at an iron-concentrating mill gave me confidence that I was on the right track. My aim was a settling box with uniform cross-section for settling, from which the settled material could be removed with a minimum amount of

slime-bearing solution without disturbance of the surface near the overflow point.

I also felt that it was necessary to have no submerged parts moving on each other to cause undue wear.

A mechanical rake that would carry the sand up an inclined trough and discharge it above the water level occurred to me as able to fill these requirements and yet give ample space for quick settling at the surface while allowing the maintenance of an agitated zone at the bottom to prevent the slime settling with the sand.

After considerable investigation the motion desired was secured by means of suspending the rakes from cross bars carrying rollers traveling on suitable tracks, with switches, as shown in Fig. 1. The rollers passed under the switches on the up stroke, but were lifted on the down stroke and thus raised the rakes over the sand which had been moved forward.

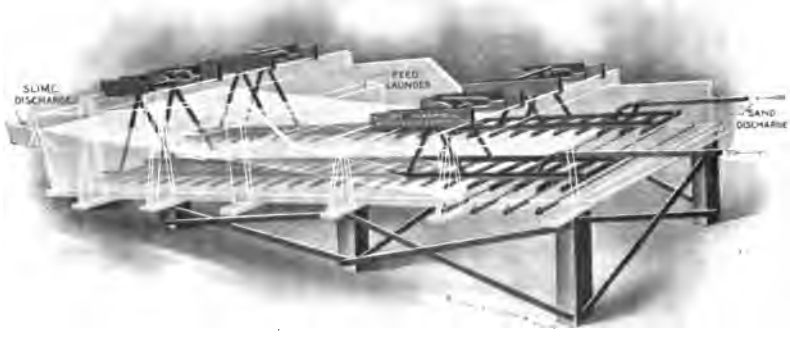

FIG. 1.—THE DORR CLASSIFIER (PATENTED). ORIGINAL DESIGN.

The installation of the machine thus designed made such an improvement in both sand and slime products that our extraction was greatly increased and lost time on account of classification was eliminated. I may say, as a matter of personal interest, that it made the difference between the failure and success of a mill that had a subsequent useful life of nine years and treated profitably ore from which not over $2.50 per ton was recovered.

Mechanical sand separation is a commonplace thing to-day in cyaniding and may have been used then in concentration and other work. It is of interest, however, to note that one of the best-known engineers in this country told me in 1906 when he saw the machine working, on his return from South Africa, that if he had seen the blueprints of it he would not have believed it possible that it could give the products it was then making.

Description

The Dorr classifier has been improved from time to time to meet new conditions and as now manufactured is shown in Fig. 2, which illustrates the standard duplex machine.

It consists of a settling box, in the form of an inclined trough with the upper end open, in which are placed mechanically operated rakes or scrapers for the purpose of removing the quick-settling material from the open end. Each rake is carried by two hangers, one at the sand-discharge end suspended from an arm attachment to a rocker arm or lever, which terminates in a roller. The other depends from a bell crank connected by a rod to the same rocker. The roller is pressed against a cam on the crank shaft.

The rakes are lifted and lowered at opposite ends of the stroke by the

FIG. 2.—THE DORR CLASSIFIER (PATENTED). MODEL "C."

action of the cams transmitted through the rocker arms and bell cranks, and the horizontal motion is obtained directly from the crank.

The bell cranks at the slime end are carried by a second larger bell crank held in position by a chain attached to a spool on a worm gear at the head end of the classifier. By this means the rakes can be raised 10 in. at the lower end and operated in that position or any intermediate one. This allows the classifier to be started readily when nearly filled with sand after an unexpected shutdown, and the regulation of the depth of the settling box when in operation to vary the products being made.

The pulp is fed across the settling box as shown and a uniform flow to the lip at the end is maintained while the sand settles to the bottom and is advanced by the rakes until it emerges from the liquid and is discharged with from 20 to 30 per cent. moisture.

The agitation near the bottom of the tank, caused by the reciprocating motion of the rakes, assists in keeping the slime in suspension, but is not normally sufficient to cause fine sand to overflow.

Regulation of Products

The machine is intended of course to make only two products, and the point of separation can be varied by the following means:

1. The use of the baffle shown, which allows a reduction of the cross-section of flow so as to overflow more fine sand.
2. Raising the rakes and so operating them in a shallower tank.
3. Increasing the speed of the rakes until the agitation keeps fine sand in suspension.
4. The attachment of perforated spray pipes to the rakes at a point where they remove the sand from the liquid, allowing a rewashing of the sand and removal of any slime that may be carried down with it.

Cost of Operation

As with any other machine the costs of operation will depend on the work done and care taken by the mill men.

The only place where wear caused by the sand would be expected is on the bottom edge of the scrapers, and careful measurements have shown that in more than one case four years' constant service did not remove more than $\frac{1}{8}$ in. at this point. This rather surprising result must be due to the scrapers being suspended and the lower edge pushing grains of sand ahead so as to prevent the bottom coming into actual contact with the sand bed.

It is not unusual for machines to run for a year or more without repairs of any sort being needed, and I know of one classifier which ran for nearly four years with repair costs under $5 and a careful examination at the end of the period showed it to be in good condition. I believe an allowance of 0.05c. per ton treated for repairs will usually be ample.

Power

The power required will depend on the load, but tests have shown that $\frac{1}{4}$ h.p. is generally sufficient.

Capacity

The capacity of the machine depends, as would be expected, on the nature and dilution of the pulp fed to it and the point of separation desired. The table below showing results at different plants will give the best idea of it.

Operating Data on Dorr Classifiers

Mill	Tons per 24 Hr.	Feed				Sand			Slime		
		Dilution	+100	+200	−200	+100	+200	−200	+100	+200	−200
Portland, Colorado	145[a]	5 to 1	38.3	13.4	47.1	75.3	16.2	7.4	0.9	5.8	94.7
Golden Cycle, Colorado	151[a]	3.5 to 1	45.0	19.0	36.0	68.0	23.0	9.0	.....	11.0	89.0
Cia. Benef. de Pachuca, Mexico.	144[b]	4 to 1	42.7	25.1	32.5	43.6	40.1	16.8	0.1	3.8	96.8
Tonopah Extension, Nevada..	73[c]	6 to 1	65.0	11.1	23.9	70.0	18.3	11.7	0.6	8.5	90.9
Amparo Mining Co., Mexico	125[a]	7.5 to 1	52.4	18.5	29.1	71.4	27.4	1.2	.....	6.5	93.5
Tonopah-Liberty, Nevada....	85[c]	7 to 1	30.5	41.9	26.3	75.7	24.0	0.1	2.5	14.3	81.4
Goldfield Consolidated, Nevada.	264[d b]	3 to 1	40.0	12.0	48.0	74.0	19.0	7.0	2.6	19.5	79.1
Tonopah-Belmont, Nevada...	70[c]	.....	65.6	2.7	28.9	.....	.....	.....	0.4	6.6	93.0
Alaska-Treadwell, Alaska[e]....	89[c]	.....	48.7	41.5	9.8	48.7	41.5	9.8	.....	2.0[e]	98.0
Nipissing, Canada	.....	.....	78.7	3.6	16.8	85.8	7.2	6.9	.....	0.5	99.5[f]

a New ore.

b Includes tube-mill return. Under normal conditions, with a tube mill in closed circuit with a Dorr classifier, 35 to 50 per cent. of original feed returned.

c New ore; tube-mill product returned, but amount not stated.

d 168 tons come from the stamps, the rest being returned from the tube mill.

e Classifiers used in connection with the regrinding of concentrates; rakes operated 24 strokes per minute; 100 per cent. of concentrates pass 200 mesh; the 2 per cent. is silica from tube-mill pebbles.

f Of this 200-mesh product 15 to 20 per cent. is sand; crushing is done in two sets of tube mills; the sand from the first classifiers being reground in closed circuit with tube mills and other classifiers, the combined slime overflows giving the screen test shown.

Uses

The Dorr classifier was originally designed for the separation of clear slime and a leachable sand, but with the adoption of so-called all-sliming

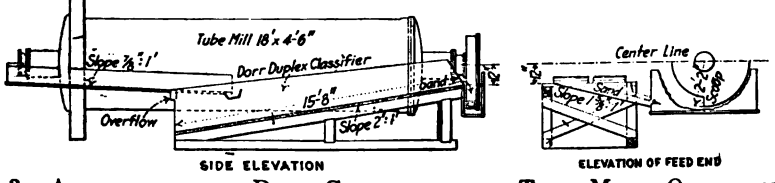

FIG. 3.—ARRANGEMENT OF DORR CLASSIFIERS AND TUBE MILLS OPERATING IN CLOSED CIRCUIT IN THE TONOPAH-BELMONT MILL.

treatment in cyaniding its value in connection with regrinding in tube mills was apparent to all and it came rapidly into use for that purpose also. The advantage of operating a classifier and tube mill in closed circuit with no other means of elevation of the return is appreciated by all practical men. Fig. 3 shows the arrangement of tube mills and classifiers as installed at the Tonopah-Belmont mill.

The use of a Dorr classifier with two or more compartments for washing the sands was early suggested and a double-washing classifier for

giving sands an acid bath was put in use at the Morning mill with tube flotation several years ago.

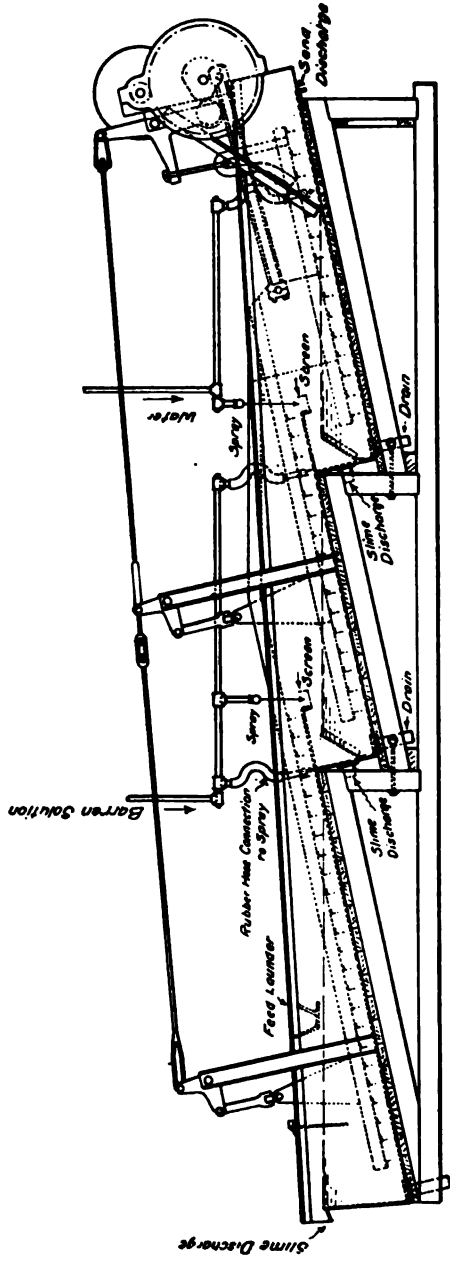

FIG. 4.—THE TRIPLE-WASHING DORR CLASSIFIER.

We have carried the idea further and made several triple-washing classifiers for use when sands are to be discharged from gold-bearing cyan-

ide solutions without leaching and must be freed from cyanide and dissolved gold.

Fig. 4 shows the triple-washing Dorr classifier with pipe connections arranged so as to wash the sands as they come from the first two compartments with barren solution, and from the last with water.

One of these machines following a duplex Dorr classifier has been installed recently at the Hollinger mill to wash the concentrates after they have been reground and cyanided in a tube mill.

The high-grade solution overflowing from the first machine assays $65 in gold per ton and the concentrates are finally discharged with 20 per cent. moisture which assays 16c. per ton; thus giving a dissolved loss per ton of concentrate of only 4c. About 30 tons per day are thus treated.

In 1910 we furnished a series of classifiers to a Pennsylvania company for the purpose of leaching and washing, by the counter-current method, cupreous pyritic residues that had been given a chloridizing roast, but I understand that other matters interfered and their use for that purpose was never thoroughly tested.

In 1913, Captain Wolvin, of the Butte-Duluth Co., conceived the idea of acid leaching and washing oxidized copper ores in Dorr classifiers from watching them at the Butte Superior mill, and at his request we furnished him with five classifiers for trial and later two more for the same series with all parts subject to contact with the acid made of hard wood. These machines proved most successful, increasing the extraction over the former leaching in vats at coarser mesh at least 20 per cent., and as a result the Butte-Duluth Co. has added to its equipment five heavy Dorr classifiers 8 ft. wide by 30 ft. long which I understand will be operated in series and have an estimated capacity of 400 tons.

I believe the results of the excellent work done by Captain Wolvin and Mr. Sherwood will be embodied in a paper to be presented to the Institute.

Dorr classifiers have been successful as dewaterers in handling magnetic iron and sulphide concentrates, and are discharging the former with only 12 per cent. moisture.

The Dorr Thickener

History

In the summer of 1906 I was engaged by the Mogul Mining Co. to change the Kildonan mill at Pluma, S. D., from a 100-ton dry cyanide plant into a 300-ton wet-crushing mill following the practice at the Lundberg, Dorr & Wilson mill previously mentioned.

The settling of slime in cyanide work in the United States at that time

was an intermittent process usually carried out in flat-bottom tanks, but a few engineers had followed the practice introduced, I believe, by John Randall, of building very large cones and operating with a continuous overflow and intermittent or continuous underflow. We had used a 22-ft. and a 20-ft. cone at the Lundberg, Dorr & Wilson mill and appreciated greatly the advantage of continuous thickening, so desired to use the same method at the other plant. The Kildonan had been started as a chlorination mill in 1894, but in 1902 the property had changed hands and as cyanide was proving successful on Black Hills ores the chlorination barrels were scrapped and replaced by some leaching tanks to treat a 20-mesh product. A little later the company was induced to install an aerating leaching process in which the bottoms of the leaching tanks were covered with perforated pipes and sand was to be leached and agitated at the same time. As might be expected, it was not a success, but added several 35 by 5 ft. tanks to the equipment.

When I undertook to remodel the mill, as before mentioned, there was no room in it to place the number of large cones that would have been required, and, as I had experienced the difficulty of preventing the building up of solids on a 60° slope when making a thick underflow, I sought some other means of thickening continuously. I conceived the idea that it might be possible to operate a mechanism or scraper in a tank at such a speed that it would not disturb the upper part of the tank and interfere with settling and yet would prevent the slime from becoming solid and allow it to be discharged at one point. It was recognized that if the slime did become solid no mechanism could be built strong enough to move it. The design of the machine presented a problem, in the solution of which I was greatly assisted by L. B. Eames, now superintendent of the Goldfield Consolidated mill, who was an assistant on construction. We decided to utilize one of the 35 by 5 ft. tanks mentioned, after deepening it to 12 ft. Realizing the immense leverage of scraper arms on any shaft, we considered a track on the periphery with a rotating frame traveling on it from which would depend the rake arms, but finally decided to try the central drive. It was recognized here as in the classifier that stuffing boxes and wearing parts in the pulp should be avoided, and the present arrangement, which seemed the simplest, was adopted.

The use of a 35-ft. tank for the first test was certainly experimenting on a large scale, especially as the first trial thickener represented the only equipment provided in the plant for that purpose. I was prepared, if it had not worked, to use the tank as a settler in another way, which I felt would give as good results as were obtained in practice elsewhere

We had done considerable good work with diaphragm pumps at Terry, and had found them capable of pumping anything up to small monkey wrenches. I had planned, therefore, to place one or more above the tank and provide each with a number of suction pipes, reaching to the bottom

and so spaced that the tank would become virtually a collection of small cones to be pumped from in rotation.

The thickener started off, however, with no trouble and we found that the first speed selected, 1 rev. in 20 min., was satisfactory. It was operated until the plant burned in 1912.

Continuous thickening with Dorr thickeners is accepted nearly everywhere now, but its possibilities were not recognized at first. When the first thickener had been operating at the Mogul for several months, a Colorado engineer, manager of one of the best operated companies in that State, happened to see it. After watching it for some time he was so confident it could not be used for continuous work on his extremely light colloidal slime that he waited nearly a year before trying it. When finally installed it showed an apparent saving of over $70 per day in increased recovery and diminished costs.

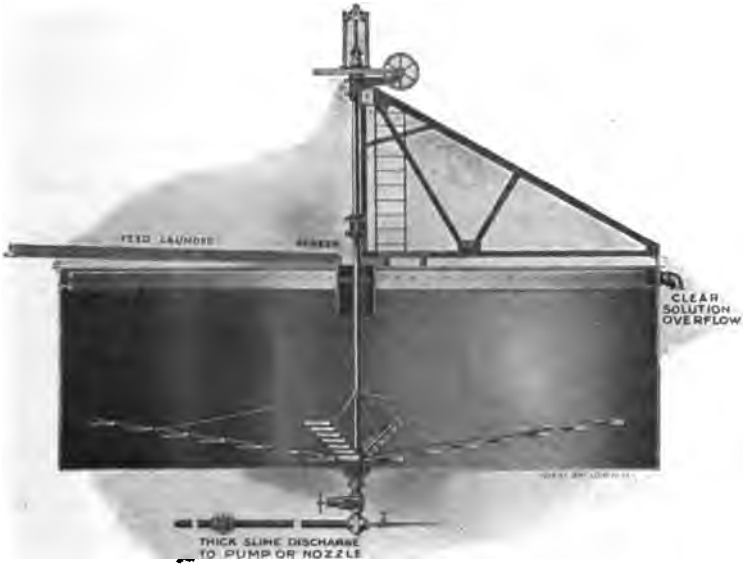

FIG. 5.—THE DORR CONTINUOUS THICKENER (PATENTED).

When we came to manufacture the machines commercially many improvements were made, the most important being the addition of an overload alarm to give warning when some part of the mechanism was in danger of being strained by the resistance offered by the accumulation of thickened material in the tank.

Description

The Dorr continuous thickener consists of a slow-moving mechanism placed in a suitable tank, by means of which the operation of settling may

be made continuous through the removal of the settled material to a point of discharge and the prevention of its accumulation as a solid in the tank.

As usually furnished, it consists of a central vertical shaft with radial arms equipped to bring the thickened material to a discharge opening at the center by the slow rotation of the mechanism. The thick material may be discharged by gravity at this point into a launder, or piped to the side of the tank and raised by air lift or pump to the level of the overflow or higher.

The machine is arranged for raising the shaft so that the arms will not be imbedded in the thick material if the power should be shut off for any length of time. The shaft can be lowered again gradually while running. Shaft and gear bracket are supported by a bridge over the tank or suspended from the roof trusses (Fig. 5).

In some cases, notably at the Liberty Bell mill, thickeners have been installed driven from below the tank through a mercury bearing, and proved efficient, although they cannot be raised.

The thin pulp is delivered near the center of the tank in a suitable well with a float to cause minimum disturbance, and the overflow is taken off by a peripheral launder.

The thickened pulp can be accumulated and withdrawn at intervals or a continuous discharge maintained as desired. Nozzle discharge is in use in some concentrating mills where a comparatively thin pulp is desired, and also in one case where a product of only 30 per cent. moisture is being obtained. Many are using diaphragm pumps for this purpose. They have the advantage of ready regulation and require little attention. Having a positive displacement they tend to regulate automatically the amount of solids withdrawn, for, when the pulp becomes thicker, more solids are pumped with each stroke.

The diaphragms have a life of from two to eight weeks, depending on operating conditions, so that renewal expenses are very small. Although arrangements have been made to change the speed and stroke of the pumps, a pet cock to admit air into the suction has proved most satisfactory as a means of regulation.

Speed

The thickener has been operated at speeds ranging from 1 rev. in 2 min. to 1 in 40 min. A quick-settling sandy material will offer great resistance unless a comparatively high speed is maintained, while colloidal slime will give a slightly thicker underflow at a very slow speed. It will be found when handling sandy material that if the sand accumulates so that it is being moved around the tank by the channel arms as well as being advanced toward the center by the plows the resistance increases rapidly and the speed should be increased.

Power

This varies with the size of the tank and the nature of the feed. Of course, the motor input is much larger than the actual power consumed, owing to the low load factor commonly used, as it is essential to have power enough to meet an unusual strain. Spring measurements have shown approximately $\frac{1}{10}$ h.p. being transmitted to the worm shaft of a 44-ft. thickener handling a classified slime. It would not be advisable, however, to install less than a 1-h.p. motor on a single machine, but $\frac{1}{2}$ h.p. each can be allowed when several are driven from one lineshaft.

Repairs

Normal operation of the Dorr thickener causes no wear except on the worm, and many machines are running to-day that have not cost a cent for several years. On the other hand, if started after a shutdown without

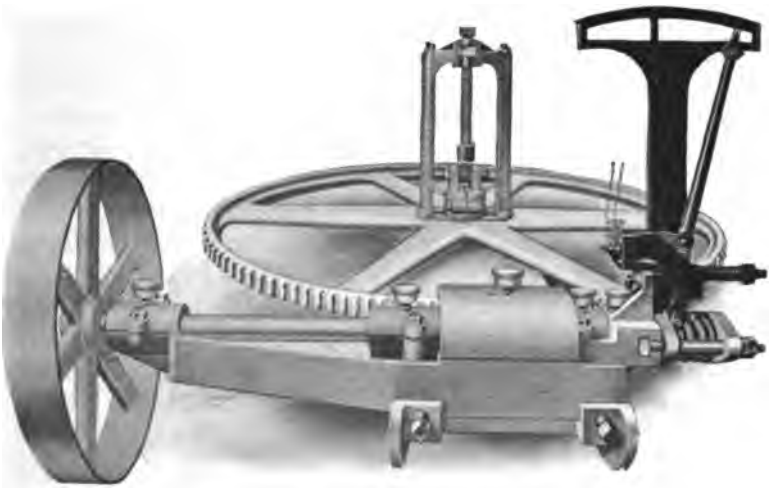

FIG. 6.—OVERLOAD ALARM AND RESISTANCE INDICATOR ON DORR THICKENER, WITH ARRANGEMENT FOR RAISING SHAFT WHILE OPERATING.

raising, a strain of any amount may be given so that the "weakest point," which I believe exists in everything but the "One Hoss Shay" of Oliver Wendell Holmes, will have to yield. We have attempted to avoid breakage by furnishing a pulley that cannot transmit enough belt pull to do any damage, but the average mill man is likely to use a crowbar to start something without realizing that he can thus apply more force than the belt.

The overload alarm, mentioned before and shown in Fig. 6, is arranged to indicate the resistance offered by the mechanism as shown by the thrust

on the worm shaft and to ring an alarm when the load becomes excessive. A solenoid or other means can be used to correct automatically the condition causing the excessive load, by reducing the feed or increasing the underflow. The alarm has proved very valuable, especially on quick-settling pulp, when it is desired to obtain the thickest discharge.

Labor

The attendance required varies with the regularity of conditions maintained and is usually confined to lubrication once a shift, so that the care of the thickeners is included in the duties of some man employed principally on other work. At the Anaconda four men per shift are more than enough to care for 160 tanks thickening 26,000,000 gal. per day containing 2,500 tons of solids.

Capacity

The capacity of Dorr thickeners has been found to be primarily a function of area, although the depth of the tank has an influence depending on the dilution of the feed and the dilution of the underflow desired. With a given area and depth and a very dilute feed and underflow, the capacity depends on the amount of liquid that can be clarified; *i.e.*, additional solids, but no additional liquid, can be added to a tank already fed to capacity without overflowing slime. On the other hand, with a feed perhaps 8 of liquid to 1 of solids and a thick discharge of 2 to 1 or less, it will be found usually that additional liquid can be added to a thickener operating at capacity without overloading it, but any addition of solids will cause slime to overflow.

If a plant requires more settling capacity, raising the temperature of the solutions may prove an economical way to add 10 to 20 per cent. The capacity of any filter will also be increased and a higher extraction may pay for the cost of heating.

The table following represents data given me from time to time. It shows the settling area in use per ton in different mills which in many cases were not feeding their thickeners at capacity.

Density of Underflow

This depends on the nature of the pulp to be settled and the size of the particles. An argillaceous pulp, such as that at the Liberty Bell in Colorado, although containing a large percentage of reground siliceous material will not settle thicker than 60 per cent. moisture, while a finely ground quartz will give as low as 27 per cent. moisture. At the Porcupine-Crown plant, handling a quartz product of 75 per cent. −200 mesh, the

average final pulp discharged contained 30 per cent. moisture when the feed was at the rate of 1 ton of solids per day for each 4.7 sq. ft. of tank area.

Operating Data on Dorr Thickeners

Mill	Sq. Ft. Settling Area per Ton of Solids Thickened per 24 Hr.	Sq. Ft. Settling Area per Gallon Overflowed per Minute	Remarks
San Rafael, Mexico...	4.5	.....	Tube-mill product, 75 per cent. − 200 mesh, discharge 45.5 per cent. solids.
Liberty Bell, Colorado	15.0	12.6	Tube-mill product, much light argillaceous slime. Discharge 33 per cent. solids: + 100, 17 per cent.; + 200, 13 per cent.; − 200, 70 per cent. Feed 9 : 1. Solution fed at capacity; solids not. Large area per gallon overflowed per minute due to density of underflow and nature of the slime.
Mogul, South Dakota	3.92	.....	Tube-mill product; ore siliceous: + 60, 0.6 per cent.; + 100, 7.8 per cent.; + 200, 26 per cent.; − 200, 65.6 per cent. Discharge 56 to 59 per cent. solids. Continuous decantation.
Batopilas, Mexico.....	0.6 to 0.9	.....	40-mesh product; 90 per cent. passing 100 mesh.
Zambona, Mexico.....	3.1	.....	Tube-mill product. Discharge 40 per cent. solids.
Dominion, Ontario....	5.4	.....	Tube-mill product, 88 per cent − 200 mesh, ore diabase. Discharge 40 per cent. solids. Feed 6 : 1.
Porcupine-Crown, Ontario.	4.25	.....	Tube-mill product, 75 per cent. − 200 mesh. Discharge 65 per cent. solids. Quartz ore. Continuous decantation. With 5.1 sq. ft. settling area per ton settles to 71 to 73 per cent. solids.
El Palmarito, Mexico	4.5	.....	Tube-mill product: pure quartzite, 97 per cent. − 200 mesh. Feed 7 : 1. Discharge 65 to 70 per cent. solids. Continuous decantation.
Amparo, Jalisco, Mex.	4.9	1.4	Tube-mill product, siliceous: 93.5 per cent. − 200 mesh. Feed 24.5 : 1. Discharge 23.5 per cent. solids: used to feed vanners.
Veta Colorado, Parral, Mex.	5.0	3½[a]	Tube-mill product, rather argillaceous: 71 per cent. − 200 mesh. Feed 11 : 1. Discharge 33 per cent. solids for agitator. Have settled to 65 per cent. solids.
Smuggler-Union, Telluride, Colo.	.....	.....	Very clayey slime with classified sand. Screen test: + 40, 1.48 per cent.; + 60, 7.27 per cent.; + 100, 14.81 per cent.; + 200, 11.63 per cent.; − 200, 65.81 per cent.
	30.0	26.0	Settling from cold water, slightly alkaline. Feed 8 : 1. Discharge 50 per cent. solids, 1.429 sp. gr.
	10.0	.....	Settling from cyanide solution. Feed, 2.5 : 1. Discharge 40 per cent. solids, 1.316 sp. gr.
A large copper company, Arisona.	11.6	8.11	Considerable argillaceous slime. Feed 10.4 per cent. solids. Discharge 25.3 per cent. solids.
Pennsylvania Steel, Lebanon, Pa.	14.2	2.48	Thickening ahead of vanner concentration. Feed 2.8 per cent. solids. Discharge 10.6 per cent. solids. Overflow 0.4 per cent. solids, extremely fine, which does not interfere with using water again.

[a] Not up to capacity of overflow.

Operating Data on Dorr Thickeners (Continued)

Mill	Sq. Ft. Settling Area per Ton of Solids Thickened per 24 Hr.	Sq. Ft. Settling Area per Gallon Overflowed per Minute	Remarks
Nevada Consolidated, Ely, Nev.	.....	1.25	"Each 17-ft. thickener supplies wash water for 20 Wilfley tables and occasionally for wash on vanners. One thickener has a greater capacity than twelve 8-ft. cones." Area of 17-ft. tank is 226 sq. ft.; of the twelve 8-ft. cones, 525 sq. ft.
Broken Hill, Proprietory, Australia.	.....	1.80	Dewatering slime from lead-zinc concentration mill. Feed 100 : 1. Discharge 55 per cent. solids.
Anaconda Copper, Mont.	.....	5.95	Dewatering slime from concentrator. Forty 4-deck thickeners, each 28 ft. in diameter by 3 ft. 3 in. deep, handle about 26,000,000 gal. of pulp per day which contains approximately 2 per cent. solids. A clear overflow obtained, the underflow containing about 15 per cent. solids, which is fed to buddles.

The data given here show that when pulp is carried in cyanide solution a provision of 5 to 6 sq. ft. per ton for a siliceous tube mill product is ample and from 7 to 15 sq. ft. for a clayey material or classified slime product. When very dilute products are handled the area required is determined usually by the gallons per minute to be overflowed.

Reground pulp, when the original material is not homogeneous, such as an ore containing quartz and some argillaceous material, may be found to have what D. L. H. Forbes has well called a "critical density." If we attempt to thicken it below this density a segregation may occur and the more siliceous portion of the pulp be discharged while an accumulation of fine slime takes place until it overflows, even though the feed is greatly reduced.

An increase of several per cent. in the moisture of the underflow will cause a return to normal conditions and the thickener will have again its usual capacity.

These conditions have been observed at the Hollinger mill at Porcupine, where a mixture of quartz and schist is milled, but they do not appear to exist at the Porcupine-Crown. I have seen some indications that the slow motion of the arms in the thickener may be of actual service in securing a thicker discharge than could be obtained by undisturbed settling.

Uses

The Dorr thickener was orginally introduced in cyaniding for thickening slime or reground pulp previous to agitation and filtration and came into general use for that purpose. The advantages of continuous thickening were enhanced when continuous agitation was shown to be profit-

able and the use of a continuous filter made a complete process where actual operating labor was reduced to a very small amount.

Continuous Counter-Current Decantation.—This method for the recovery of pregnant solutions was first operated in this country, I believe, by John Randall in the Black Hills in 1901 with the large cones heretofore mentioned. Being familiar with his work and the difficulties met in maintaining continuous operation in cones, I recognized in 1907 that the thickener would make the process feasible, and designed the first plant to use it in 1910. This plant has been in successful operation up to the time of writing.

Since then over 20 plants have been erected or are now being installed to operate by this method. Several of them are replacing filters of different types with results apparently satisfactory to their owners, although no detailed figures have yet been published.

The method is especially adapted to the treatment of quick-settling pulp requiring a weak solution for the dissolution of gold and silver. Where pulp cannot be thickened to 50 per cent. moisture or less, or where a strong solution of cyanide is required, it will usually pay to install a filter after the decantation system to reduce the mechanical loss in cyanide and at the same time the dissolved losses which will then be higher.

The subject of continuous decantation is too large to be discussed in detail in this paper, but I hope that a paper will shortly be presented to the Institute covering actual results obtained and discussing it in theory as well as in practice. The advantages of its use in the leaching of copper ores with acid solutions is apparent, as was demonstrated by Dr. Rudolph Gahl several years ago in a test plant at the Detroit Copper Co. I believe that considerable test work is now being done on the same line.

Concentration.—The use of the thickener in concentration followed shortly after its wide adoption in cyaniding. The Nevada Consolidated Co. was the first of the large copper companies to use it in place of cones and obtained good results, as is shown by the following quotation from a letter from the mill superintendent there:

" . . . One thickener, 17 by 9 ft. deep, has a greater capacity than 12 8-ft. cone tanks, for the reason that there is only 1 unit to watch instead of 12, without the constant result of feed to 12 cones becoming unbalanced. Also the one 8-in. spigot used on the Dorr requires no attention compared with the dozen $\frac{3}{8}$-in. spigots on the cones. . . ."

It will be noted that the large difference in capacity per square foot is attributed to advantages in regular operation. Under *test* conditions no advantage has been found in a comparison of the thickener with 8-ft. cones on dilute pulps.

The recent installation of the Anaconda Copper Co. will be described, I understand, at this meeting.

The development of the flotation process has created a further demand

for the machine and it has come into extensive use at Broken Hill, and in most of the plants in this country using that process.

The idea of many able concentrating men that the capacity of a settler is in proportion to its peripheral overflow rather than its area, appears, in fact, to have no good foundation where large-sized units are used for completely settling out all solids. One reason for this, I think, is that in such work it is common to have a varying amount of clear liquid always on the surface of the tank. When thickening pulp in an alkaline solution, as in cyanide work, where there is a sharp line between a coagulated slime of definitely heavier specific gravity and clear solution, it is often found that a thickener will have nearly the same capacity if the overflow is taken off by a pipe at one point, instead of by a peripheral launder carefully leveled. With the above conditions there will be a flow of liquid directly from the feed box in the center to the overflow pipe at one side. This will naturally tend to raise the slime level in that path above that in the remainder of the tank, and as a result, the slime-bearing liquid will sink as if it were a non-miscible liquid of greater specific gravity, and the clear liquid from all over the surface of the tank will flow toward the outlet pipe in its place. Samples taken at varying depths have shown almost equal density at equal depths at points in the path of such a flow and at the opposite side of the tank.

I do not wish to be understood as recommending the omission of the overflow launder; for when a tank is crowded to its limit it always allows more leeway before the slime overflows, and many operators have found considerable increase in capacity by its use, but cite this as bearing on the relations between capacity and area.

Caetani has discussed the matter very thoroughly in the *Mining and Scientific Press* of Mar. 22, 1913.

Recent work has shown the advantages of thickening both ahead of, and after flotation to the greatest density possible, using the warm overflow of the final thickeners to dilute the underflow of those ahead of flotation to the proper density for that process. This means conservation of a large percentage of the heat required and also a large saving in oil needed. One plant now following this procedure has found that, whereas laboratory tests indicated that over $\frac{1}{2}$ lb. of oil was required per ton, the actual consumption does not exceed $\frac{1}{8}$ lb.

Industrial Uses.—The thickener has been used successfully in rubber reclamation to settle some finely divided residues and it would appear that there should be many opportunities for its use where operations are on a large enough scale to make the advantages of continuous over intermittent work apparent. At one plant its introduction saved the labor of 12 men.

Acid-proof Thickeners.—The absence of submerged wearing parts has made the construction of acid-proof thickeners a comparatively simple

matter and we have manufactured them for the Chiquicamata plant of the Chile Exploration Co., and for the Butte-Duluth Copper Co.

The Dorr Tray Thickener

The tray thickener was designed by me in 1913 for the purpose of securing large settling area in a comparatively small space. I have not wished to make the invention widely known until we have had ample experience with actual operation under various conditions. Its development has been slow, owing to my desire to determine by actual installations, its limitations, advantages, and the simplest methods of construction.

Description

The basic principle lies in the use of a plurality of superimposed shallow settling areas operating with a common mechanism either in

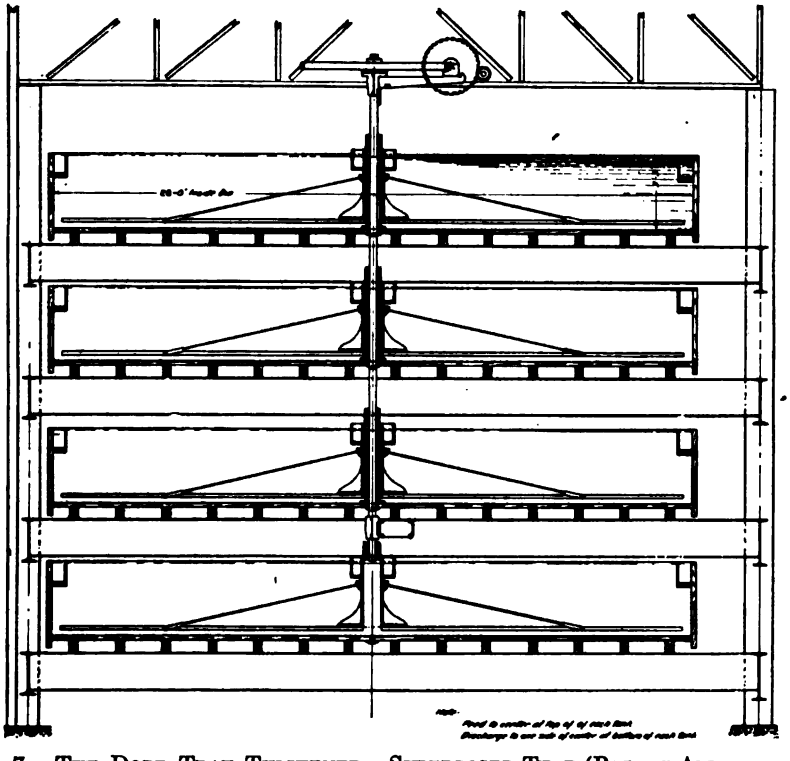

Fig. 7.—The Dorr Tray Thickener. Superposed Type (Patent Applied for).

separate tanks, as in the installation at Anaconda, Fig. 7, or with submerged trays in a deep tank, as at the Homestake, Tomboy, and Liberty

Bell. The former method presents the advantages of accessibility, but the latter requires less head for a given capacity and gives no loss of

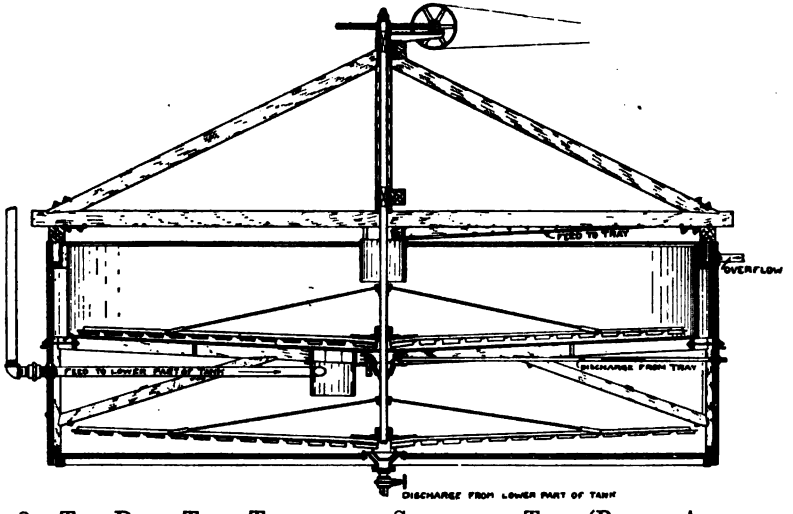

FIG. 8.—THE DORR TRAY THICKENER. SUBMERGED TYPE (PATENT APPLIED FOR).

mill height for the overflow. It has the merit of offering a simple means of increasing the capacity of any thickener in operation without the need

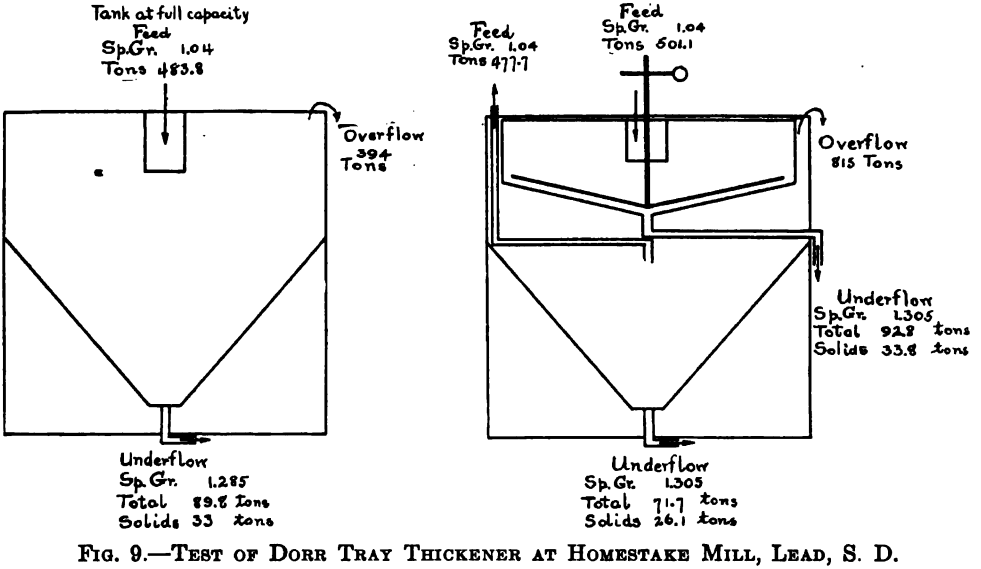

FIG. 9.—TEST OF DORR TRAY THICKENER AT HOMESTAKE MILL, LEAD, S. D.

of adding building, foundations, superstructures or materially changing launders, etc. We have only gone as far as adding one tray at present,

in the case of the submerged trays, but have found no reason why two or three should not be used.

Fig. 8 shows our latest type of tray supported, as will be noted, both from the side and bottom of the tank. A cup attached to the rotating shaft carries a rubber gasket which makes contact with the small cone in the center of the tray and prevents passage of the slime to the space below. The thick-slime discharge is carried out at the side of the tank at present. Each settling space has an independent feed but the combined overflow of both is usually taken off at the rim of the tank. It has been found unnecessary to make the tray strong enough to support itself when full with the tank below empty, as an automatic float is provided to connect the tanks in an emergency.

The first test made on this type was at one of the settling plants of the Homestake Mining Co. through the courtesy of A. J. Clark, metallurgist in charge, and gave the results shown in Fig. 9. As will be noted, the tray was added to a cone-bottomed tank and the depth is greater than usual. It showed an increase in capacity of approximately 90 per cent.

The first tray to be installed in a Dorr thickener, already in operation, was at the Tomboy last fall and since then they have been added at the Liberty Bell and Tonopah-Belmont. We have also furnished more to the Tomboy.

Capacity

Careful tests at the Liberty Bell, where a very good form of automatic distributor of pulp is in use, indicate that under conditions there, the tray placed in a 33 by 10 ft. thickener has added 100 per cent. to its settling capacity. The tests mentioned and those made by the Anaconda Co. indicate that the additional capacity to be obtained by the addition of the tray will depend upon the relative depth and discharge of the tank in which it is placed, the dilution of the feed and the underflow, and the nature of the material settled. It appears conservative to estimate that an addition of from 70 to 95 per cent. can thus be made. It must be recognized that a deep tank has a definite reserve capacity which is valuable when the feed and settling qualities of the pulp vary. If such variations occur during each day a thickener 12 ft. deep might need only the area required by the average feed, while one 3 or 4 ft. deep would have to be able to handle the maximum feed during a period and so might require considerably larger diameter. Mr. Waddell, manager of the Ohio Copper Co., has told me that the variations in the slime at the Nevada Consolidated were so great that tanks 20 ft. deep were warranted as equalizers. This plant now has eight tanks 50 ft. in diameter by 20 ft. deep besides 13 smaller ones.

The greatest field for the tray thickener is where tanks have to be

housed and space is valuable; or heated pulps have to be settled and a minimum of radiation is required.

The Dorr Agitator

History

The Dorr agitator was designed in 1910 for a mill that I was then planning, but as it was decided to omit agitation at first, it reached the Patent Office before receiving an actual test. The first work done with it on a small scale was by Noel Cunningham, now superintendent of the Hollinger mill, who at that time was doing some test work for Mr. Jones of the Bel-

Fig. 10.—The Dorr Agitator (Patented).

mont, and the results obtained showed an average of nearly 4 per cent. better extraction than was made by the Pachuca tank against which it was tested.

The first machine to operate on a commercial scale was installed at the mill of the Hollinger Mines, Ltd., to handle an extremely quick-settling siliceous product which would stall the agitators originally used in 36 hr. It proved entirely successful and was operated for four months when a

change to partial continuous decantation made direct agitation unnecessary and it was replaced by a thickener. The second installation of machines proved able to handle a 30-mesh product successfully.

Early in 1913 agitators were installed to handle concentrates at the high-grade mill of the Buffalo mines, at Cobalt, and they have given continuously good results on this most difficult work.

Description

The Dorr agitator as now made consists of a vertical pipe suspended in the center of a tank carrying at its lower end hinged sweeps with

FIG. 11.—THREE DORR AGITATORS, 32 FT. DIAMETER BY 15 FT. DEEP, AT BUCKHORN MILL, NEVADA.

scrapers adapted by their rotation to bring any settled material to the pipe. The upper end of the pipe terminates in a head casting carrying two distributing launders and supported by a short vertical shaft with driving mechanism carried by the roof trusses of the building or a special truss over the tank.

A worm and sheaves provide means for raising or lowering the hinged arms at will. Air is introduced into the lifting pipe either at the bottom or by means of a pipe through the shaft, which is then made hollow.

Fig. 10 shows the machine and the method of operation. The Dorr agitator may be briefly described as a Dorr thickener, with only two hinged arms, and a central air-lift pipe with distributing launders substituted for the central shaft. Fig. 11 shows three of these machines operating in the Buckhorn mill, Nevada.

Speed

The speed can be regulated to suit conditions. It is usually operated at from 1 to 3 rev. per minute, depending on the size of the tank and nature and density of the pulp being agitated.

Power

The mechanical power taken varies from ¼ to 2 or 3 h.p. according to the size of tank, speed, nature and dilution of the pulp handled. The air required is from 8 to 40 cu. ft. per minute at 10 to 20 lb. pressure.

Tables below show data as given us by some of the engineers using the agitators. It will be recognized that it is difficult to measure small quantities of low-pressure air and mechanical power and to secure satisfactory data on the operation of individual machines.

The following covers what we have been able to secure on the operation of our agitators:

Data on Operation of Dorr Agitators

Buffalo Mines, Cobalt, Ont.

Agitators on Concentrates in High-grade Mill

Tank, Feet	Speed, Rev. per Min.	Dilution	Cu. Ft. Free Air per Minute
20 by 12	10	1 : 2.8	28
10 by 10	7	1 : 2.8	..
10 by 10	9	1 : 2.8	..
10 by 10 *	3	1 : 2.8	..

* Using worm-gear drive.

Mechanical power not measured; all machines giving good results.

Agitator on Slime from Low-grade Mill

Tank	Dilution	Cu. Ft. Free Air per Minute
16 ft. 6 in. by 19 ft.	1 : 1	40

The air measurements on these agitators were carefully made and are stated to be accurate within 1½ per cent.

Nevada Hills, Nevada

Size of agitator, 30 by 14 ft.; speed, 6 rev. per minute; capacity, 80 tons solids at 1 : 2½; semi-siliceous silver ore, 70 per cent., 200 mesh; mechanical power, 1.4 h.p.; air, 50 cu. ft.

Remarks: "Agitation good; no dead areas; no cost for repairs in 1½ years; practically no attendance."

Liberty Bell, Colorado

Ore largely silver, 17 per cent. +100, 70 per cent. −200 mesh; dilution 1 : 2.15.

Size of agitator, 33 by 11 ft. Air that passes a ⅛-in. hole in $\frac{1}{16}$-in. lead gasket at about 19 lb. pressure.

Size of agitator, 17 by 17 ft. Cone bottom (formerly used for Hendryx agitator). Air passing through $\frac{3}{16}$-in. hole in gasket; mechanical power about 0.4 to 0.5 h.p. at 8 rev. per minute.

Remarks: Results of five tests showed practically no difference in extraction as compared with the Hendryx.

Tonopah Mining Co., Nevada

Ore, silver, 75 per cent. −200 mesh; dilution 1 : 1.

Size of agitator, 36 by 20 ft. Total power for air and mechanical operation estimated at 5 h.p.; speed, 3 rev. per minute; power estimated for paddle agitator with four air lifts previously used, 17 h.p.

Remarks: "Results satisfactory in every way."

Mogul Mining Co., South Dakota

Siliceous gold ore, 60-mesh product, 7 per cent. +100 mesh; 64 per cent. −200 mesh. Dilution 1 : 1½.

Size of agitators, 16 by 16 ft. Air that passes $\frac{3}{16}$-in. diaphragm at 15 lb. pressure; power at 2 rev. per minute, measured by spring transmission to bevel-gear shaft, $\frac{1}{20}$ h.p.

Buckhorn Mines Co., Nevada

Ore, gold, very clayey.

Motor input on three 32 by 15 ft. agitators and lineshaft, 2.82 h.p.; actual output (allowing for power factor) 2.13 h.p. Agitation satisfactory; extraction reported to be better than tests indicated.

Ophir Mines, Colorado

(Not in operation)

Ore, gold; 75 per cent. −200 mesh, somehat clayey; dilution, 1 : 1.

Size of agitators, 16 by 16 ft. Air, 8 to 10 cu. ft.; about 20 lb. pressure; speed, 1 rev. per minute; mechanical power under ½ h.p.

These agitators ran satisfactorily on 30-mesh stamp product.

Renewals

Any machine of this type can be broken when started with the pulp packed if the arms have not been raised. It has, however, stood up

under very hard usage and demonstrated that with very ordinary care there should be no breakage even with quick-settling material. All parts subject to a normal wear will last several years.

Efficiency

The value of an agitator depends on the cost of making the maximum economic extraction by its use. This will be determined by its cost of agitation per ton per hour and the time required to obtain dissolution.

All the data I have been able to obtain indicate that the cost of agitation per hour is extremely low. Extraction tests that have been made indicate that the rate of dissolution on both gold and silver ores in Dorr agitators, with a uniform although less violent agitation, is at least as rapid as in any other agitators with which it has been compared.

The most favorable conditions for the dissolution of gold and silver in cyanide solution vary with each ore treated. The maintenance of an excess of dissolved oxygen throughout the whole mass of pulp and the free movement of all particles of solids in the liquid immediately adjacent seem to be the only conditions that can be generally specified.

Definite work both in milling and the laboratory indicates that many ores will give a more rapid extraction and allow the use of a weaker solution when agitated as a dilute pulp of 3 to 1 or 4 to 1, and also cause less chemical consumption of cyanide.

The Dorr agitator with its combination of air and mechanical agitation gives a flexibility that is apparent and it insures keeping *all* the solids in suspension all the time whether the pulp is subjected to a gentle or a violent movement. This is essential especially in continuous agitation.

It will thus be seen that the intensity of agitation can be regulated to suit the chemical needs of the ore being treated, rather than the mechanical necessity of keeping it in suspension in all parts of the tank.

Selective Agitation

The question of selective agitation was discussed by Mark R. Lamb[1] in the *Transactions* of the Institute. As used here the expression is taken to mean continuous agitation under conditions which cause the coarser particles of the ore to remain in the agitator longer than the average time of treatment and thus give them the longer exposure they may require to obtain the best extraction.

This can be accomplished readily with the Dorr agitator by agitating at a dilution which allows the coarser material to classify out and discharging the raised pulp near the center while the outflow is taken from near the periphery. It will be seen that if a segregation takes place and

[1] *Trans.*, xl, 775 (1909).

the agitator is fed a pulp carrying 10 per cent. plus 100 mesh, the discharge may be only 5 per cent. plus 100 mesh at first. With twice as much inflow as outflow of 100-mesh material it will concentrate in the agitator while gradually increasing in the outflow until an equilibrium may be reached in which the feed and discharge will both be 10 per cent. and the average pulp in the tank 20 per cent., so that the exposure of coarse sand would be approximately double the average. While this is theoretical, enough work has been done to indicate that the idea can be put to practical use.

Conclusion

In closing this description of Dorr equipment I wish to express my indebtedness to many friends in charge of the operation of plants using it, who by their constructive criticism and willingness to suggest and try out new ideas have greatly assisted me in making the many improvements that have been added from time to time and overcoming weak points as they have developed.

The classifier, for example, was designed originally to handle 100 tons of ore per day; but more work has been thrust upon it until at the Goldfield Consolidated it has taken care of 260 tons daily, including tube-mill returns. The demand of A. H. Jones, of the Tonopah-Belmont mill, that it be made capable of acting as a sump to take the drainage of the tube mill led to the designing of the present type, Model C, with rakes that can be readily raised 10 in. at the lower end.

I am particularly indebted to D. J. Nevill, of Stearns-Roger Manufacturing Co., who is our consulting mechanical engineer and who is largely responsible for the mechanical development of the present machines.

Ancient Auriferous Gravel Channels of Sierra County, California

BY MARK N. ALLING, DOWNIEVILLE, CAL.

(Salt Lake Meeting, August, 1914)

INTRODUCTION

A FEW years ago, gravel mining in the ancient river beds of Sierra county was the principal industry of the county. Operating drift mines were very numerous and many large hydraulic mines were in action. Eventually the State laws put an end to hydraulic mining, it being claimed that the silt carried down the streams to the valleys filled the rivers and caused winter floods, which were damaging to the surrounding farms. The ancient river deposits which could easily be found were, in the course of 40 years, worked out or practically so. The prohibition of hydraulic mining was a severe blow to the industry in general. The deposits were not developed and the gravel-mining industry practically stood still for many years. In Sierra county, nearly all the hydraulic mines were at the points where the ancient channels were exposed to view by reason of the modern stream erosion, or, in the miner's language, where the channels "enter" or "break out of" the ridges. The claims covering them also covered large areas in which were long segments of lava-capped channels, suitable for drift mines, and the points being hydraulicked were usually the controlling points of attack. Men who have followed hydraulic mining for any length of time seldom are content to work a drift mine. Knowing the great value of their deposits and feeling that they had been robbed by the State, these men have stubbornly held to their claims. The gold that is carried in these gravel channel deposits is more or less scattered throughout the depth and width of the deposit; the percentage that can be recovered by the process of drifting ranges from 50 to 75 per cent. of the total. In hydraulicking, practically all of the deposit is put through the sluices, and as the cost per cubic yard is only about 10 per cent. of the cost of drifting, the deposits, if classed as hydraulic mines, are much more valuable than if classed as drift mines; hence the old timers have held their claims (now worthless as hydraulic mines) at prices which the drift-mining industry could not stand; and such mines

are still intact. Gravel deposits formerly valued at a great many thousands of dollars have reverted to the government. Some of the claim owners, realizing the truth, have sold their mines for reasonable figures, and many of these properties are now being reopened.

But there is another and very important source of future drift mines. Only the deposits which could easily be discovered have been found. There are many unworked segments of ancient channels in this rich district; some have been discovered in a general way but never exactly located or traced through the country; still others have been partly opened up, requiring the investment of more money than the owners possess. The district is isolated, timbered and brushy (especially in the vicinity of the moist gravel deposits), and hence the more difficult forms of prospecting have been left to the coming generations.

The long period of comparative inaction in the gravel-mining industry has given rise to the common opinion that the gravel deposits are all worked out; but such is not the case. The field is young and large, and yet to the stranger passing through the county for the first time the impression is given that the district is an old and exhausted one. This is principally due to the fact that in the boom days, when the shallow placers were yielding fortunes for the thousands who worked them, large towns sprang up, only to be deserted in whole or in part upon the exhaustion of the shallow placer deposits and the desertion of the hydraulic mines.

Following the exhaustion of the discovered placers, and the suspension of hydraulic mining, came the search for the sources of the gold. Mining companies began to reopen the old drift mines, exploring for the quartz ledges, and with such remarkable results that the annual gold output of the county was increased last year by about $1,250,000. Other companies are now exploring for the undiscovered channel segments.

Historical Geology

The Ancient Drainage System of California

It is claimed by the best geological authorities that at the end of the Tertiary period the entire country now occupied by the Sierra Nevada mountains had been leveled down into the form of a great plain (peneplain). The streams were filled with quartz gravels, almost to the exclusion of softer rocks, and these streams were rich in gold. The original surface undoubtedly contained much more quartz than is evident upon the general surface to-day. The quartz, being the hardest and most resistant rock of the district, was battered about through time, and while vast quantities of it were ground to sand, enough survived to fill the streams as gravel, while its softer associates were converted to sediments and washed into the valleys. Even at the present time, in certain high

localities where the sheltering lavas have but recently been removed, we find wide zones containing large ledges and massive bodies of solid quartz.

The flow of these rivers was interrupted, first by an unknown agency and later by the intermittent flows of three distinct types of lava.

That there was an interruption prior to the coming of the lavas is evident from the fact that we find two systems of channels, running in parallel courses through the same ancient valleys, and both of prevolcanic age; the one system being secondary to the other yet both buried by the same initial ejection of acid lava ash. This condition has come under my observation, and I find that H. W. Turner suspected something of the sort when studying the gravels of northern Sierra county, 20 years ago.

The blocking of the original channels may have been due to one of three causes:

1. (*a*) To an uplift of the coast line, having the effect of lessening the grades and thus causing the channels to choke with débris and forcing the waters to seek new courses; or (*b*) to local undulatory movements of the earth's crust. As we have some direct evidence of such movements, cause (*a*) seems more probable than (*b*).

2. To débris washed into the rivers during great storms which may have accompanied or immediately preceded the long period of volcanic disturbances.

3. To a light ejection of volcanic ash, temporarily drying up the rivers and causing the waters to choose new courses, although still confined to the same ancient basins or valleys; the nature of this material being such that eventually it was washed away, leaving no boulders or other easily recognized evidence of its visit. Thus the "unknown agency" may really have been a very premature warning of the approach of the volcanic age.

Regardless of the cause, there was a long uninterrupted period of stream action following the end of the more ancient system of prevolcanic rivers, during which period the secondary channels of this age had time to cut their new beds, from a few feet to 100 ft. or more below the level of the originals.

While the lava outbursts were intermittent, and this was true to a greater degree in some districts than in others, the entire country, in time, was covered with three general types of lava.

1. The rhyolite, a light-gray, fine-grained, acid lava ash. This was the first lava to be thrown out, and, as compared with the other lavas, in very limited amounts.

2. The andesite, a heavier, semi-basic, darker, mottled lava. It was ejected first as tufa and mud flows, mixed more or less with the rhyolite, then as breccias, and later as massive flows of heavier and darker boulders and mud, semifluid in consistency. It flowed into the depressions, ravines, rivers and valleys.

3. The basalt, a dark, almost black, heavy, basic flow mass, in places

so fluid that upon sudden cooling a natural glass (obsidian) was formed. This lava was the last to be extruded, pouring out as a capping over the others.

The darker the lava, the more basic, and the more basic the more fluid, so that the basalts flowed the farthest from the craters.

The rhyolite was washed into the river beds from the hillsides, covering the gravels and forming what is known to the California miners as "pipe clay." On the top of this lie the andesitic tuffs, breccias, and muds with the hard resistant basalt forming a protecting roof to the ancient river deposits, and where the river valleys were not entirely filled this took the form of long, narrow, level surfaces, conforming to the sinuosities of the underlying channels. In some portions of the State subsequent erosion of the adjacent, softer country rock has resulted in leaving these lava-covered areas as long sinuous mesas, standing high above the surrounding country, so that the ancient channels have been little disturbed by modern erosion. In all parts of the State this condition existed up to a certain late period; but in the northern portions subsequent lava flows have brought about a different effect; the modern streams having cut to pieces the ancient system.

The action of the craters was intermittent. In Sierra county there were at least three long periods of rest between the outbursts, during which times the streams were endeavoring to reinstate themselves.

It appears that the longest rest periods, with the exception of the last one, were at the beginning of the disturbances, for we find that the intervolcanic streams of the first and second rest periods usually flowed in the same ancient valleys occupied by the original rivers and on one side or the other of the same, while it is not uncommon to find a secondary channel on each side of an original. Their courses were governed by the fact that the valley was not filled with the lava; the heavier lavas sought the center of the depression, and being more viscous, upon cooling, formed a harder and more resistant capping than the lighter, looser and drier material that had been crowded to the edges. Due to this same weakness, this material along the edges was more easily eroded by the waters rushing from the hillsides, so that the new intervolcanic channels are found skirting the edges of the original channels, or crossing them now and again. The point of crossing is usually at the place where two branches of the original river came together, or where there is a sudden turn in the original river. The course of any stream generally changes at the point of confluence of a large tributary; the extra width of the valley at such points would allow the cooling lava to settle more than in the narrow places, and the new channels, having a higher and wider valley to begin with, would naturally flow straight until turned by the opposite bank.

During each period of rest there was evidently some segregation of

materials in the craters, for preceding each extrusion of heavy lavas there was an ejection of more or less ash. This rule of action seems to have been followed until well along toward the last of the volcanic period; for with the exception of a few of the latest intervolcanic channels we find, in all of these ancient channels, some kind of an ash, covering the gravels, and in various stages of decomposition. The later the age of the channel the darker and usually the coarser is this ash, and, so far as my observations go, the quantity is also decreased.

A white pipe clay does not cover the gravels of an intervolcanic channel; neither does a dark, flinty, granular or "chocolate" pipe clay cover a prevolcanic channel. It was long held that the color was due to chemical changes after the ash was ejected, brought about through alteration of the constituent minerals under the action of the infiltrating waters, etc., but, in my judgment, this is not the case. The color and texture, in most cases, depend upon the relative age in which the ash was ejected (although of course alteration has taken place in some localities), the acidity depending upon the period, and the color upon the acidity.

Relative Economic Value of Ancient Channels.—The history of drift mining has proved that these ancient channels are valuable in the following order:

The old, original, prevolcanic, high-quartz gravel channels are always rich; likewise the secondary, prevolcanic, quartz gravel channels. The intervolcanic channels of the earlier periods are generally rich, depending upon their position relative to the prevolcanic streams and upon the character of the country rock. The third and fourth age intervolcanic channels are very uncertain, as their values depend principally upon their association with the older channels, having had very little chance to collect minerals from the underlying bed rock. The fifth-age channels, or modern streams, were rich, having obtained their gold from the long stretches of ancient channels cut away and from the ledges eroded below the level of the same, as the modern streams have enjoyed a very long period of rest, and have worn their way down more than 2,000 ft. into the mineralized country rock below the level of the ancient streams.

The Uplift of the Sierra Nevada Mountains.—The most widely accepted theory, among modern geologists, is that during the volcanic period, or immediately following it, the entire eastern portion of what is now California was elevated many hundreds of feet. The granites and other igneous intrusives forced upward the overlying sedimentary rocks, the country faulting along a north-and-south line approximately the same as that which now marks the boundary line between the States of California and Nevada; and the old peneplain, with its lava-filled rivers, was tilted to the westward on about a 2° slope.

The effect has been that the original grades of the southwest-bound rivers have apparently been increased by about 1 per cent., or about 50

ft. per mile; so that channels which originally flowed on a grade of say 100 ft. per mile are now found beneath the lava with a grade averaging 150 ft. per mile. In localities where the channels flowed northerly the opposite effect took place.

The country, however, was elevated *en masse*, so that the ancient channels still lie in their proper relative elevations and positions. Local faulting has been discovered in southern Plumas county, but in the Sierra county drift mines little or no faulting of a serious nature has been met with.

Following the uplift, the modern rivers flowing westerly cut their way down through the lavas and old channels and deep into the country rock. This erosion has been a rapid one, as the streams flow in deep narrow gorges, and the work of the waters was no doubt aided by the action of glaciers.

The modern rivers which have effected the Sierra county district are the middle and north branches of the Yuba river, with their numerous tributaries.

The Economic Geology of Sierra County

Sierra county is about 55 miles long, east and west, and about 20 miles wide. The summit of the Sierra Nevada mountains passes through the central portion of the county in a northerly and southerly direction. All the mines lie on the western slope of the mountains.

In the extreme western part of the county the rocks are principally black, highly metamorphosed clay slates, designated by the government geologists as the "Delhi slates." They are underlain by granites which are exposed in the bottoms of the deep canyons. Porphyry and diorite dikes are common in the slates. A few good mines have been found in this formation.

Lying just east of the Delhi slates is a zone about 10 miles wide, consisting of alternating belts of "Calaveras" formation[1] and serpentine

[1] The "Calaveras formation" is a general term applied to the older, or Carboniferous, sedimentary rocks of the "Bed Rock" or "Auriferous Slates" series. It is composed of seven different types of sedimentary rocks, which have been given names indicating the locality in which they are most common:

1. The Clipper Gap formation; composed of clay slates, cherts and limestones.
2. The Delhi formation; black siliceous rocks, rarely schistose.
3. The Cape Horn formation; fissile clay slates.
4. Relief formation; cherts and quartzites.
5. Blue Canyon formation; fissile clay slates, quartzitic sandstones, cherts, and limestones.
6. Blue Canyon formation; contact-metamorphic rocks, chiefly mica schists.
7. Limestone lenses; in Carboniferous and Juratrias formations.

While the Dehli slates are themselves included in the Calaveras formation series, the rocks most generally called the Calaveras formation by the miners in the Forest City district, are the sedimentary schistose rocks; often referred to simply as "slate."

or amphibolite. This zone parallels the course of the Sierras, flanks the range, and is the line of weakness. It is highly fractured and mineralized and forms what is known as the "Father lode" of California, in contradistinction to the "Mother lode," which occupies a similar position relative to the range in Amador, Calaveras, Tuolumne, and Mariposa counties. In this zone are found most of the quartz mines and practically all of the gravel mines.

Next to the east lies a zone of quartz porphyries, granites, slates, and occasional small dikes of serpentine. Here are found the Sierra City quartz mines, some of which have been large producers. The zone is from 3 to 5 miles in width.

Between the Sierra City quartz-porphyry zone and the summit few mines of importance have been found. Beyond the summit no mines are found until the State of Nevada is reached.

The Father Lode, or Contact Zone.—The Calaveras slates of this zone strike north and south and are tilted at a high angle to the east. The amphibolite schists and serpentine intrusions range in width from a few feet to 2,000 ft. These lenticular bodies of serpentine strike northerly and southerly, paralleling the general strike of the bedding planes of the sedimentary rocks. Some of them extend through the district for many miles; others are but short bodies.

The slates through which these intrusions are found have been thoroughly shattered, tilted, altered, and in some cases highly metamorphosed, especially near the contacts. The zone has been so thoroughly fissured and mineralized as to cause the government geologists to make special mention of that fact.

Quartz ledges are commonly found following directly along the contacts, lying in the schists or slates and paralleling the contacts or extending from the contacts out into the slates as true fissure veins. The shattered slates are full of quartz lenses, pockets, and veins of variable size and richness, but all carrying gold enough to hold the attention of the prospector.

In a general way these long, parallel, hard serpentine lenses are separated by the alternating belts of the softer and heavily mineralized slates, and here the ancient streams have cut their beds. In wending their way south and west, they seem to have experienced considerable difficulty in crossing the serpentine bodies, and often followed along the east side of the serpentine until the end of the lens was reached, when the channel would swing to the west until another serpentine body was encountered. Where the channels did cross the serpentine the gold did not lodge, as a rule, while the uneven, decomposed surface of the slates seems to have formed an ideal gold-catching bed rock. Nearly all of the ancient rivers of this district followed this mineralized zone. The gold and other heavy minerals lodged in the beds of these streams.

No doubt the greater part of the placer gold came from the general erosion of the great quantities of mineralized material, but the very large pieces of gold found all along in the "Great Blue Lead," which passes through this district, evidently came from the ledges. Many of these ledges were exposed in the workings of the gravel mines, and several old mines have been reopened and developed as quartz mines; this proving that the large gold slugs do not travel far from their sources.

The lava mantle which covered this country still lies several hundreds of feet deep on the ridges, hiding the contacts and ledges and rendering prospecting slow and costly.

The early drift miners paid little attention to the quartz ledges; they knew little about handling the ore and they were not prepared to do so. The gravel was rich and they were making money. They did not care to divert their energies and, as they expressed it, "waste their time in trying to get it out of the hard rock." By simply washing the gravel in their sluices they could gather from the riffles the clean gold dust and nuggets. So the ledges were passed and many of them forgotten, and yet the memory of some of them has led to important discoveries in the present day.

Rich ledges were found in the gravel diggings of the Tightner, Red Star, Rainbow, Sixteen to One, Dead River, South Fork, North Fork, Bald Mountain, Ruby, Telegraph and many other drift mines.

In the Bald Mountain mine a ledge was discovered which has given rise to many "fairy tales" of great riches, and upon investigation I have secured enough reliable data to cause me to believe that success would follow the reopening of the tunnel for the exploration of this ledge. Success in this undertaking is still more assured from the fact that about 1 mile of unworked channel awaits development in this ground and could be reached by extending the tunnel which was run for the quartz. This ledge is from 2 to 3 ft. in width. It crops up through the bed rock and into the gravel from 1 to 2 ft., and strikes northerly and southerly, crossing the channel diagonally, since the channel was there coursing southwesterly. The ledge was followed in the breasts for from 300 to 400 ft., and the gravel just below the crossing was the richest of any worked in the 7,000 ft. of channel exhausted by the company. A large number of chunks of gold were found, many of them still being partly imbedded in the quartz matrix.

Sulphides were found in the gravel at this point and interfered with the sluicing; they were shoveled out as a nuisance. It was not until recent years, when so many rich strikes were being made in the neighboring ledges and the gold found was so often associated with arsenical sulphides, that it has dawned upon the owners that these sulphides were probably valuable.

The Bald Mountain Co. bought 9,600 ft. of Oregon creek as a tailings claim after that creek had been completely worked out by the pioneers.

A number of nuggets, varying from 30 to 100 oz., some still clinging to the quartz matrix, were found in the creek when the company was reworking the tailings. These had been carried through the sluices with the coarser gravels or had been forked out with the boulders.

The ledge itself was considered a nuisance by the miners, as it stood up in the gravel and interfered with the breasting and timbering.

The Bald Mountain and Ruby mines were noted for the great amount of coarse gold contained in their gravels. This feature of the channel which flowed through them did not cease when the Bald Mountain was passed; but the channel continued to yield large rough nuggets in the claims lying south of the Bald Mountain and even as far south as Alleghany and beyond. These nuggets did not all come from the Bald Mountain and Ruby claims, by any means, but were liberated from the ledges followed and crossed by the channel.

The Tightner, Red Star, Rainbow, and Sixteen to One have been reopened and now rank as quartz mines of value; the Tightner mine at the present time is probably paying the biggest dividends of any mine in the State. In the last year it has produced about $1,000,000 in gold. From December, 1912, to July, 1913, it yielded in round numbers, $700,000; in July, 1913, about $50,000 was taken from a small stope 9 ft. square.

The South Fork, North Fork, Dead River, and Telegraph are now being developed for their quartz ledges.

A local company is making preparations to reopen the Ruby and Bald Mountain mines this summer, both for the ledges and for the gravel which they are known to contain.

To the north of the Ruby, and below the lava cap, the Downieville-Forest City contact has been traced down through Slug canyon, through Downieville, and up the next mountain side to the lava cap again, a distance of about 6 miles. The Loosner group, Carson group, Tripple Pocket group, Frost claim, York mine, Donderro claim, Bessler mine, Good Hope group, Oxford mine, Bösch claim, and Standard mine group cover this entire distance. Coarse gold has been found all along this contact from Forest City to the Standard mine, a distance of about 8 miles, and some of the above-named claims and mines have produced large sums. Slug canyon, which follows this contact for about 2 miles, derived its name from the character of its gold. This same contact passes through the Ruby mine and the Bald Mountain mine, through Forest City and on south to Alleghany. The Tightner mine lies due south from the Bald Mountain ledge, and the Tightner ledge strikes due north, but as the intervening country is lava capped it is impossible to say that the Bald Mountain ledge is the Tightner ledge.

In the Ruby mine the ancient channels followed this contact, and on the old company maps may be seen the words "Line of coarse gold," in-

dicating the approximate line along which the large slugs were found. Pieces ranging from 10 to 45 oz. were commonly found here and one nugget was found which weighed 201.56 oz. A model of this nugget is in the museum of the California State Mining Bureau. We have record of at least 12 nuggets ranging in weight between 30 and 170 oz. found in the South Fork or Live Yankee mine, just south of the Bald Mountain. Large gold slugs were found in the Tightner gravel, and after a lapse of 40 years the mine was reopened. The ledge proved to be richer than could reasonably be expected of any ledge. Adjoining the Tightner is the Sixteen to One mine, where the gravel also yielded large specimens. The ledge was rediscovered a short time ago and considerable gold has already been extracted, the last strike (a $30,000 pocket) being made about six months ago.

Just south of these properties the Rainbow gravel yielded large slugs. One quartz boulder was found which contained $36,000 in gold. The ledges were found in the breasts. After 30 or 40 years this mine was reopened and something over $300,000 was taken from the quartz ledge; one slab of quartz being found which was calculated to contain $20,468 in gold.

The Oriental mine, situated on a serpentine contact about 1 mile west of the contact just described, has produced about $3,000,000, and it is commonly reported that one pocket yielded $740,000. The Plumbago mine, about 3 miles to the southeast, has produced about $2,000,000, of which $250,000 came from one stope. The Brush Creek mine, situated on the Mountain House serpentine contact, about 4 miles to the west, yielded $400,000 from one shoot. The Tripple Pocket mine, on Slug canyon, and on the Forest City-Downieville contact, and the Gold Bluff mine, near Downieville, were heavy producers. The Sierra Del Oro or Old Ironsides mine lies about 2 miles to the east of the Ruby mine and is the latest big strike in the district; the ledge is just being developed, but several thousand dollars have already been recovered from specimen ore.

The Downieville-Forest City and Alleghany contact line has proved to be the longest, most productive, and most constant and continuous gold producer of any contact line in the entire county.

The Ancient Channels of Sierra County.—The ancient channels of Sierra county all flowed southerly, being northern tributaries to the main ancient river which flowed westward in Nevada county from North Bloomfield to Smartsville. The rivers flowed in parallel courses and lie so close together that it would be impossible for them to have all been in action at the same time. Moreover, their relative positions, elevations, grades and characteristics mark them as having distinct periods of action.

The lava flows, which were important enough in this section to force the channels to choose new courses, evidently did not extend as far south as the great valley occupied by the main trunk stream; for that great

river seems to have maintained its position and general course, and its deposits, which are very large and deep, are not capped with lava for great distances.

Hence we find all the Sierra county channels of the various periods leading into the same ancient drainage basin as tributaries.

In the ridge north of the North Yuba river we find 10 prevolcanic channels (and at least 1 intervolcanic channel) coursing southerly and all within a section of country only 12 miles in width. Beginning on the east and naming westerly, these channels are known locally as:

Graycroft and Wide Awake.	Prevolcanic.	White and blue quartz gravels.
Excelsior and Monte Cristo.	Prevolcanic.	White and blue quartz gravels.
White Bear.	Prevolcanic.	White quartz gravels.
Telegraph.	Secondary and probably intervolcanic.	White and blue quartz gravels.
Hilo.	Prevolcanic.	White and blue quartz gravels.
Eureka.	Prevolcanic.	White and blue quartz gravels.
Brown Bear and St. Charles Hill.	Prevolcanic.	White and blue quartz gravels.
McMahon's Ranch.	Prevolcanic.	White and blue quartz gravels.
Lost River and Bunker Hill.	Early secondary.	Quartz and quartzite gravels, darker colored than above.
Brandy City and Camptonville.	A very large prevolcanic.	White and blue quartz gravel channel, extending well up into Plumas county and forming a large system of its own.

These have either been mined or are now being developed, with the exception of the Hilo channel, which is a tributary to the Eureka Lead.

There is strong evidence that the Monte Cristo channel, which is unmistakably prevolcanic, is secondary, for it appears to have cut away the higher prevolcanic White Bear channel. The channel that "enters" the ridge south of Downieville at the City of Six mine, and known as the "Deep Rock Creek channel," is a secondary channel of the prevolcanic period.

In the ridge lying south of Downieville, or north of Forest City, there are 20 ancient rivers with subordinate tributaries, all lying within a district about 18 miles in width.

Since some of the channels found in the ridge lying north of Downieville undoubtedly united in the 6 miles intervening between these ridges, they would be represented in the south ridge by a lesser number, and the writer is inclined to believe that at least 11 of the 20 channels either came from other sources, are secondary, or are intervolcanic rivers, owing their origin in part to the 6 miles of intervening territory now eroded away by the North Yuba river.

Beginning on the east and naming westerly, these channels are known locally as follows:

American Hill	Prevolcanic.	White and blue quartz gravel.
Nebraska and Mabel Mertz.	Prevolcanic.	White and blue quartz gravel.
Maple Grove.	Prevolcanic. Sec.	White and blue quartz gravel.
Bald Mountain Extension.	Prevolcanic.	White and blue quartz gravel.
City of Six and Deep Rock Creek.	Prevolcanic. Sec.	White and blue quartz gravel.
Bald Mountain Channel, or Great Blue Lead.		White and blue quartz gravel.
Deep Bald Mountain.	Intervolcanic channel of first age; at headwaters gravel is volcanic, but after leaving the Bald Mountain ground it contains so much quartz gravel that it could easily be mistaken for a quartz gravel channel.	
North Fork.	Prevolcanic.	White and blue quartz gravel.
Redding and Young America.	Prevolcanic. Sec.	White and blue quartz gravel.
Corotoman, Upper Lead.	Prevolcanic.	White and blue quartz gravel.
Corotoman, Lower Lead.	Intervolcanic.	Mixed gravels.
Lucky Dog, N. & S.	Early intervolcanic.	
Lucky Dog, E. & W.	Late intervolcanic.	
Omega (Same as Lucky Dog E. & W.).	Late intervolcanic.	
Mountain House Drift Mine.	Prevolcanic.	White quartz.
Kanaka Drift Mine.	Thought to be extension of Mountain House Channel, prevolcanic.	White quartz.
Gale.	Prevolcanic secondary or early intervolcanic.	
True Grit.	Prevolcanic.	White and blue quartz gravel.
St. Clairs.	Prevolcanic, but probably secondary.	
Snowden Hill.	Intervolcanic channel of the fourth period.	
Pleasant View and Orient.	Early intervolcanic.	Appears to have flowed northerly, being the only one of its kind in this section. Average grade is about 40 ft. per mile.
Camptonville.	Prevolcanic.	White and blue quartz gravels.

All of these have been worked as hydraulic or drift mines or are now being developed.

In the ridge lying south of Forest City these channels must all exist, although many of them have not been discovered. The ridge lying south of Alleghany contains some of them. The west end of this ridge is low, however, and some of the higher channels have been completely eroded away. As the deep canyon of the Middle Yuba lies next south of the last-named ridge, the next appearance of any of these channels is in Nevada county, where in a short distance they are merged into the great deposits which fill the main ancient drainage basin, referred to above.

Of all of the northern tributary channels flowing from Sierra county, there appear to be three main tributaries or systems within themselves:

1. The Camptonville channel heads well up in Plumas county and courses southwesterly through the western side of Sierra county, joining the main trunk at North San Juan, in Nevada county. The towns, Whiskey Diggings, Howland Flat, Port Wine, Scales, Brandy City, and Camptonville are situated along this channel. As much of the deposit was but thinly capped with lava, many large hydraulic mines operated here.

2. The American Hill channel heads in eastern and central Sierra county. The upper reaches of the channel are still undiscovered, the principal developments having been made in the vicinity of American hill, where extensive drifting and hydraulicking have been carried on. While this is not an especially large channel, it is by far the largest ancient tributary in the eastern portion of the mineralized zone.

3. The Great Blue Lead of Sierra County. This large ancient channel lies midway between the other two, heads in the high mountains of Plumas county and courses southerly through the central portion, which is also the richest portion, of the Father lode or serpentine contact zone, previously described. This main channel with its numerous large tributaries has been wonderfully remunerative. Wherever found in Sierra county it has been worked with large profits, yielding as a rule from $400 to $500 per linear foot where breasted, and is generally considered worth from $200 to $225 per linear foot in profits.

This channel derived its name from the fact that its lower strata of gravel were of a bluish color, and at the time of its discovery it was supposed to be a portion of one main ancient river extending practically the full length of the State, paralleling the Sierras in a manner similar to that of the San Joaquin and Sacramento rivers. But upon the discovery of numerous other channels of similar character and off the line of this supposed single river, this "Great Blue Lead" theory gave way to the theory of an ancient drainage system complete within itself, on the order of the present-day drainage. This general fluviatic theory has proved to be the true explanation of the conditions.

The term "Blue Lead" is misleading: It refers to the color of the material usually found in the lowest portions of the channel trough. The gravel itself is composed of white and colored quartz pebbles and boulders with a filling of sand, clay, mud, sediments, etc. The iron salts in this material have not been altered beyond the ferrous forms, as the material, due to its low position, is usually under water. The gold seeks the lowest places, and hence is usually found associated with the bluish material. The same material on the high rims or benches is white, brown, or reddish, as the iron salts have there been oxidized to the ferric compounds. When the "blue gravel" is removed from the mine and exposed to the air and light it soon changes its color, or if the water has had a chance to escape from the low portions of the channel the gravel will not be blue.

The gold may be there whether the gravel is red, white, or blue, for the color is wholly the result of the conditions which control the underground waters.

The term is usually applied to prevolcanic channels, often containing little else but white quartz gravel; it could be more fittingly applied to describe intervolcanic channels, which are usually filled with dark gravels.

Why These Deposits are Suitable for Drift Mines.—To the north, near the head waters of the ancient streams, the channels are smaller, are full of big boulders, lie on very steep grades, are more deeply and completely buried with lava, and are harder to find and more expensive to mine, owing to the high altitude, severe climate, lack of roads, etc. To the south lie the broad ancient valleys, where the channels are from 1,000 ft. to a half mile in width; the gravel is generally small, light and sandy, and the deposits are from 100 to 500 ft. in depth. The gold is fine and scattered. No definite, absolute concentration has taken place. With the widening of the channels and a decrease in the grade, came a decrease in the velocity, so that the coarse gold was dropped further back in the hills, where the rims were more pronounced and the waters more confined.

Therefore these great gravel deposits form ideal hydraulic, steam-shovel and dredging fields.

Sierra county occupies an intermediate position between the two extremities. The channels are of medium size, varying in width from 150 to 500 ft. The gold is fairly coarse, yet enough fine gold is associated with the gravels to warrant the use of quicksilver in the sluices. The concentration is good, the pay having been restricted to the one main trough and to the gravels confined to the immediate beds of the channels. The grades run from 2 to 5 per cent., so that cars may be trammed directly up the beds of the channels. The size of the gravel is medium, and the boulders are usually within the "one-man" size. The gravel is from 2 to 10 ft. thick, as a rule. The breasts are usually carried about 6 ft. high, from 4 to 5 ft. being gravel, the remainder being bed rock. All of this gravel may carry some pay, and although most of the gold lies close to the surface of the bed rock, it usually pays to wash the entire body of breasted material. These conditions, when it is considered that the entire district through which these channels have flowed is highly productive of gold, mark Sierra county as the best drift-mining field known to gold miners.

Furthermore, the facilities for mining make the field inviting. The country is well watered, and timbered, the climate is most healthful, towns and roads are built and mail and telephone service is at hand.

A great many of the channels can be traced. The segments are of good length, averaging from 2,000 ft. to 3 miles, so that the discovery of only one segment may mean a fortune. As the development involves con-

siderable expense, companies are usually organized for working these drift mines. The deposits are usually developed by means of tunnels, so that the mines are self-draining and the material can be handled by gravity methods.

Effect of Modern Erosion on the Ancient Channels.—The ancient rivers flowed southerly. The modern rivers flow westerly and have cut away many miles of the older and higher channels and collected the gold. The deep canyons in which they flow are separated by high parallel ridges. The segments of the old channels appear near the tops of these intervening ridges still buried with the lavas, but having two ends exposed where the channels have been cut away by the erosion of the modern streams, as shown in Fig. 1. This peculiar condition has given rise to the expressions that a channel "enters," "runs through," "breaks out of" or "leaves" a ridge.

To an observer standing at the bottom of one of the deep canyons the country appears remarkably broken and rugged; but from the tops of the

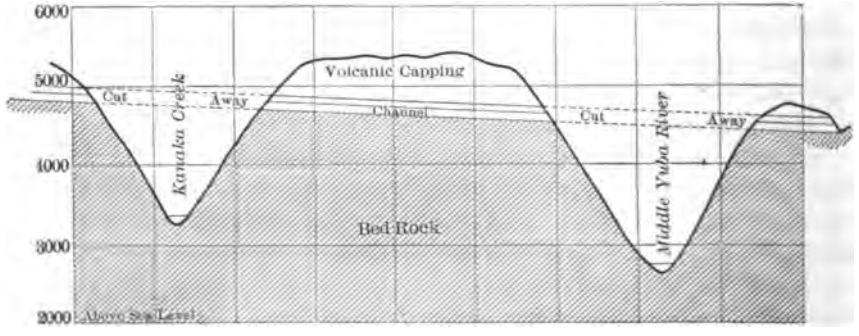

FIG. 1.—EFFECT OF MODERN EROSION ON ANCIENT DRAINAGE SYSTEM, SIERRA COUNTY, CAL.

ridges the general view is that of a vast plain, sloping gently to the westward, and into which deep east-and-west grooves have been cut.

Tracing Ancient Channels by Scientific Methods.—By applying engineering methods, many of the old channels can be traced through the country, across the canyons from one ridge to the next, by the identification of the segments found in the respective ridges.

The pioneers discovered as early as 1851 that there were gold-bearing deposits high on the mountain sides. The gold in the shallow placers was traced to these sources, but it was many years before the true cause of these deposits was understood.

To find the exact points where the channels "enter" or "leave" the ridges is often difficult, owing to the local landslides, deep loam, and dense vegetation. Wherever developed, these deposits have proved very rich. It appears that comparatively few of the ancient river segments

have been discovered and worked out; even in the mines o
many benches and tributary channels were overlooked, and i
stanc es, as in the Ruby mine, Forest City, the channel was lo
had een followed and mined for half a mile.
A s the district has been held back through lack of trans
facilit ies, it is in many respects a frontier, and the field offers
indu ements to energetic engineers and development compa
are st rong enough to carry the dead expense necessarily acco
the e tensive preliminary work.

The Value of Ancient Channels

It s impossible to get complete data on the production of
mines of California, but those which I have been able to s
consid ered reliable.

While the old channels in California have yielded from $10
profit per linear foot, where the yield for fairly long distances is
like m odern streams they have short stretches of poor pay, en
holes, and wide bars of low-grade gravel; so that before juc
passed upon the value of a deposit the work should be carried t
far bey ond that of mere discovery.

The character of the bed rock often has a great deal to do
richnes s of the gravel. Even in the Great Blue Lead, which
betwee n $400 and $500 per linear foot for many miles, a place v
on ser pentine bed rock where the channel was practically b
more t han 400 ft. In the Ruby mine, some unusually rich
found on the serpentine bed rock, so that the inference cannot
made t hat serpentine bed rock is always a sure sign of poor pay,
in the majority of cases it seems to be so. Any hard, hom
or slick bed rock is unfavorable; porphyries and granites are f
schists, slates, and especially the so-called "Calaveras format
exceptio nally good gold catchers.

While it is the general consensus of opinion that Alaska
Klondik e are very rich gold fields, it is interesting to note tha
ing to t he data collected by A. H. Brooks, the average yield of t
streams was about $100 per linear foot, and that of the White
of the K londike was only $380 per linear foot; showing that th
yield of the California channels was from $50 to $150 per line
excess o f that of the northern streams.

In a general way it may be said that the ancient channels
county yield from $150 to $500 per linear foot. The Cam
Brandy City-Scales-Port Wine channel is said to have yield
$1,000,0 00 per mile. The White Bear channel, near Downie
breaste from 40 to 100 ft. wide and yielded approximately

from 1,100 ft. The next 600 ft. of the same channel, worked by the New Monte Cristo Co., yielded about the same amount. For short stretches, this channel yielded as high as $400 per foot, and for about two years the average value of the gravel extracted by the White Bear Co. was $7 per car. The costs ran from 20 to 33 per cent. of the gross yield, exclusive of cost of plant and property. The mine had no road and all supplies were packed to the mine, on mules, and owing to the deep snow in winter nearly five months' supplies had to be laid in each fall.

The Bald Mountain channel yielded a total of $3,100,000 from 7,500 ft. of channel. The gravel in the central portion of the pay streak averaged from $3 to $7 per car load, but as the gravel could be mined and treated for from $1.25 to $1.44 per car, much low-grade ground was worked, and all material extracted from the mine, whether waste or pay, was charged against the total number of loads, bringing the general average down to about $2.17 per load.

The Ruby and the Bald Mountain Extension gravel averaged from $2 to $5 per car load, the costs of mining and washing being about $1.25 per load. The Young America gravel mine, near Forest City, yielded $7 gravel for a long working period. The North Fork mine gravel paid $5 per load for hundreds of feet. This mine was not worked out and is now being reopened.

It is usually safe to figure that each breaster will extract, on the average, about five car loads per day. The more breasters, therefore, in proportion to the outside men, the greater will be the profits, since the wage item is about 78 per cent. of the total operating expense. In the larger channels, where the development work is the most systematic, gravel averaging $1 per cubic yard, or about 70c. per car, if free, will about pay for extraction. Cemented gravels require blasting and crushing or disintegration by use of mills or revolving trommels, so that the costs of mining and treating are from 50c. to $1.50 more per ton than is true of the free gravels.

Throughout the Downieville-Forest City-Alleghany section gravel averaging from $1.50 to $2.50 per cubic yard is considered pay; but where such values are found richer gravel is usually found in the center of the pay leads.

Acknowledgments

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The views of other geologists and engineers, such as Ross E. Brown, John Hays Hammond, S. B. Christy, H. G. Hanks, J. B. Hobson, Russel L. Dunn, William H. Storms and many others, have been duly considered, and grateful acknowledgments are tendered herewith.

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The Drumlummon Mine, Marysville, Mont.

BY CHARLES W. GOODALE, BUTTE, MONT.

With notes on other mines of the Marysville district by WALTER MCDERMOTT, London, England, and F. L. SIZER, Dos Cabezas, Ariz.

(Salt Lake Meeting, August, 1914)

THE purpose of this paper is to review the history of the Drumlummon, one of the famous old mines of the West.

Mining engineers, when sent to examine new mines in old districts, or to decide whether an old property, under new conditions, will pay to reopen, have had difficulty in learning anything authentic regarding workings which are under water, and historical facts about the working character of the ore, the total production, and the value of the ore with increasing depth. Not many years elapse after the pumps are pulled, before maps, books and records are destroyed, and the value of the publications of the Institute would be increased if they contained the history of many old mines now abandoned. The unsuccessful efforts of the writer quite recently in trying to obtain reliable information about a gold mine which was reported to have produced several million dollars between 1880 and 1888, the workings of which are now partially caved and under water, impressed upon his mind the desirability of having a permanent record of old properties. In this particular case, the owners insisted that under present conditions of improved methods of treatment —particularly by the use of cyanide, which was not an applied process when the mine was operated, the property could be made to pay a handsome profit on ore left in sight in the old workings, but no assay records could be found to confirm these assurances, and even the records of bullion shipments had been destroyed. If surveys and maps were made as the work progressed, no trace of them could be found, and therefore opening of the ground through a new shaft and drifts therefrom would be attended by some risk while exploring near the old workings. Fatal accidents by flooding under these conditions have occurred, and as an evidence that miners have fears of such dangers, it was urged upon the Mining Committee during a recent session of the Montana Legislature, by a delegation of

miners, that a law should be passed compelling mine operators t
of all workings in the office of the State Mine Inspector.

Location

The Ottawa mining district is located in Lewis and Clar
tana, and Marysville, distant from Helena about 18 miles
erly direction, and having an altitude of 5,360 ft. above s
only town in the district. It is the terminus of a branch
thern Pacific Railway, which leaves the main line at Clough
west of Helena. Silver creek, which has its rise at M
through the district, and empties into Prickly Pear cre
the junction of that stream with the Missouri river. Be
ville and for 4 miles down Silver creek, the gravels of
very productive in gold, which was discovered in 1862, but
were not found until May 1, 1864. During the prospero
mining there was a lively town called Silver City, near
the diggings, and it was of such importance at one tim
records were kept there. In the meantime, Helena wa
ortance, and there was a county-seat rivalry between that
It is related that in 1865 the matter was settled when
s of Helena went up to Silver on horseback, took t
s, rode back to Helena, and depositing the books the
a new county seat.

Records of yield from placer mines are very difficult to obtai
bly safe to say that the Silver creek gravels produced $
old was not of remarkable fineness (it would sell for abou
ounce); that is, it contained considerable silver, and, in view of
g character of the ore from the principal vein of the di
Drumlummon, this is not surprising.

The gold from Last Chance gulch, where Helena is situate
about $17 per ounce, and as this gulch was producing before Sil
it is probable that the creek got its name from the more silvery
of its bullion yield.

History

The Drumlummon lode claim was located by Thomas Crus
althou h the ground had been covered by a previous location
been al owed to lapse. Cruse had been working the placer mine
creek, nd after he had opened up some high-grade ore in t
lummo claim, he built a five-stamp mill in 1880, and this m
beginn ng of prosperity for the town of Marysville. Other lo
lode cl ims were made by Cruse and by other prospectors as s

value of the Drumlummon discovery became known. Fig. 1 shows the claims in the vicinity, with the course of the principal veins.

In February, 1883, Cruse sold the Drumlummon and some small adjoining claims to an English syndicate, and the Montana Co.,

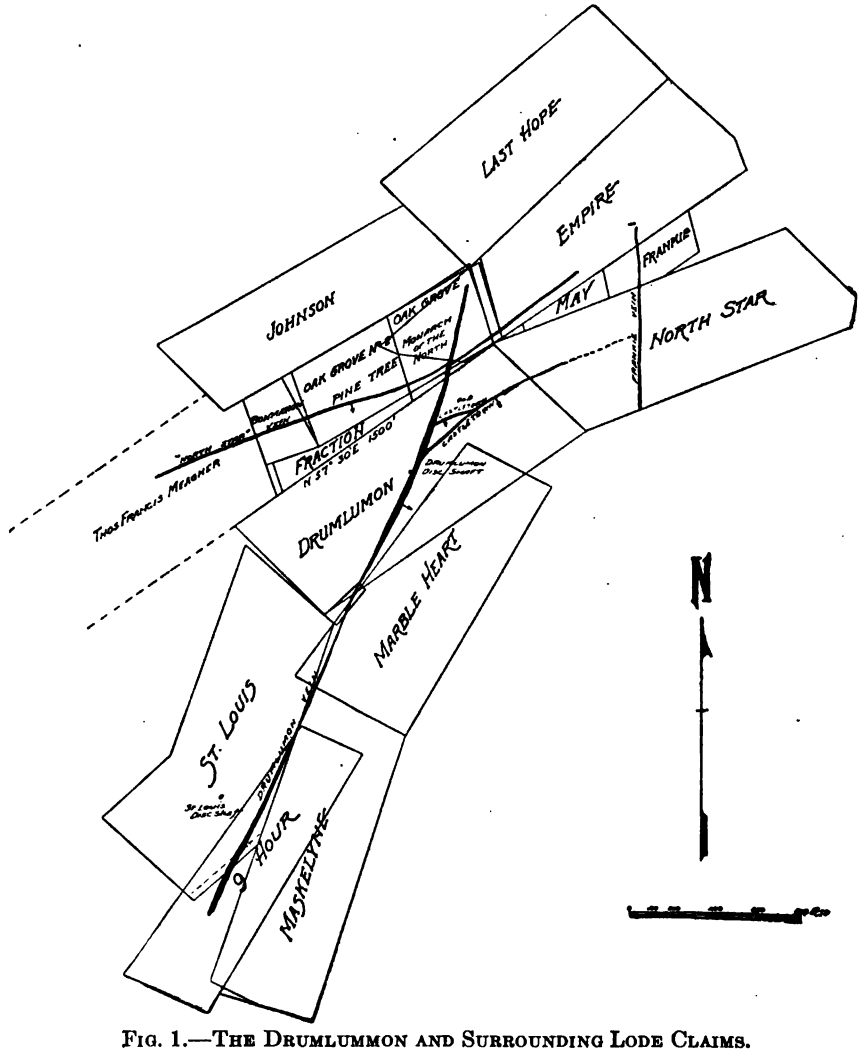


FIG. 1.—THE DRUMLUMMON AND SURROUNDING LODE CLAIMS.

Ltd., was formed under the laws of England, with 300,000 shares at £2 each. Cruse received $1,000,000 and shares worth $500,000. The report on which the mine was purchased was made by John Darlington, an English engineer, in November, 1882, and the workings at that time consisted of shallow prospect holes, a tunnel about 200 ft. long which had

been run by Cruse single handed, and which cut the vein at a depth of about 140 ft., and 361 lin. ft. of development on the vein. The orebody, as shown in a crosscut 50 ft. from the surface, was 65 ft. wide, but further exploration showed that this crosscut was run at the junction of the Castletown vein with the Drumlummon. The mine had then produced 6,000 or 7,000 tons of ore, part of which had been reduced in arrastres, but 3,780 tons had been treated in a stamp mill, and had yielded $144,539 in bullion.

Development and Exploration

The first work done by the company below the Cruse workings was to sink an incline shaft from the Cruse tunnel, and a new tunnel was started in 1883 at a point near the foot of the mountain. It reached the vein 260 ft. below the Cruse tunnel and 400 ft. below the "discovery" in a distance of about 1,260 ft., and was given the name "Maskelyne," in honor of the chairman of the board of directors. After exploring the vein for some distance northerly and southerly, a large excavation was made in the hanging wall of the vein at the end of the tunnel for a hoisting engine and headframe, and a new three-compartment incline shaft was started in the spring of 1886, having a pitch of about 68° from the horizontal. Two years later this shaft had reached a depth of 800 ft., and the vein had been opened up on the 500, 600, 700, and 800 levels, but ore values had decreased rapidly as depth was gained, and in the spring of 1888 Prof. Joshua E. Clayton was employed by the company to make a thorough geological study of the mine. In his opinion: "The impoverishment of the ore shoots below the 400 level is due to local causes rather than to permanent ones, where surface influences could not permanently affect the lode in so short a depth. . . . There is no geological reason for the lode becoming poor in so short a distance from the present surface, nor is there any chemical cause apparent for such a change in the metal contents of the lode. My conclusion is that the causes are purely mechanical and local, and not due to the or.ginal structure or other primal cause."

Sinking was resumed soon after, and by the close of 1891 the shaft, called No. 1, had reached a depth of 1,600 ft., with considerable development on the 1,000, 1,200, and 1,400 ft. levels. Connection had also been made on the 1,200-ft. level with No. 2 shaft, which was started in 1887 from the 400-ft. level, 700 ft. south of No. 1.

In the meantime the 400-ft. level had been driven several hundred feet south, and had discovered the 9-Hour shoot. Exploration of this orebody below the 400-ft. level was by means of No. 3 shaft.

From 1883 to 1910, when the ownership of the mine passed from the Montana Mining Co., Ltd., to the St. Louis Mining & Milling Co., the development and exploration work amounted to 123,500 ft., and as the

mine produced 1,150,000 tons of ore, the quantity per linear foot was not large—only 9.3 tons. This would indicate one of three conditions: First, that the veins were small; second, that the orebodies were widely separated; or third, that a great deal of unproductive work was done below and beyond the pay ore. The last condition applies to the Drumlummon, and it is doubtful if in any mine in the world the ground has been so thoroughly prospected.

Geology

This subject has been thoroughly covered by the U. S. Geological Survey[1] and will not be taken up at great length in this paper. Figs. 2, 3, 4, and 5 have been copied from Mr. Barrell's paper.

Marysville is situated in an amphitheater, with high mountains rising all around it, except on the east, where Silver creek flows out. Quartz-diorite is the bed rock of this basin, and extends up the mountain sides for several hundred feet above Marysville on all sides. Overlying the diorites are the hornstones and slaty shales, and at a point about 400 ft. above the town and southeast of it, is the "discovery" of the Drumlummon vein. This great lode, with a strike of about N. 15° E. and an eastward dip of 65° to 80°, has been opened up for a length of more than 3,000 ft. and a depth of 1,600 ft. Practically, it is entirely in hornstone, the composition of which is shown in Table I; but as it is near the hornstone-diorite contact, which has a very jagged outline, the diorite appears as vein walls at many points, where tongues of this rock extend eastward from the main body.

TABLE I.

	Barren Vein Matter 1 to 3 ft. wide Per cent.	A Foot-Wall Hornstone Per cent.	B Hanging-Wall Hornstone Per cent.
SiO_2	23	58.2	62.5
FeO	2.7	3.4	3.4
CaO	33.7	7.9	5.2
MgO	5.2	1.8	2.3
Al_2O_3	4.6	18.0	18.5

Fig. 2 shows the position of the vein in reference to the contact. Below the 400-ft. level the grade of the ore decreased rapidly in value, and while considerable tonnage was extracted below the 1,000-ft. level, it is doubtful if the bullion yield returned a profit over mining and milling cost, beyond that depth. On the 1,600-ft. level, where there is nearly a mile of exploration drifts and crosscuts, the vein which had yielded so well in upper levels gave assays of only a dollar or two in gold

[1] Joseph Barrell: Geology of the Marysville District, Montana, *Professional Paper No. 57, U. S. Geological Survey* (1907).

and silver from selected pieces. The ore occurs in distinct shoots, which have a decided trend or rake to the south. This is clearly shown in Fig. 6, with the names given to the various shoots by the management, and it will be noted that two of them do not extend to the surface—the Jubilee and Jubilee No. 2.

In width the lode varies from 15 to 25 ft. in its widest and productive portions, to less than an inch at one observed point in its pinched and barren region. For hundreds of feet along its strike it shows between well-defined walls from 1 to 3 ft. of soft material, almost free from quartz and of the character which is shown in Table 1.

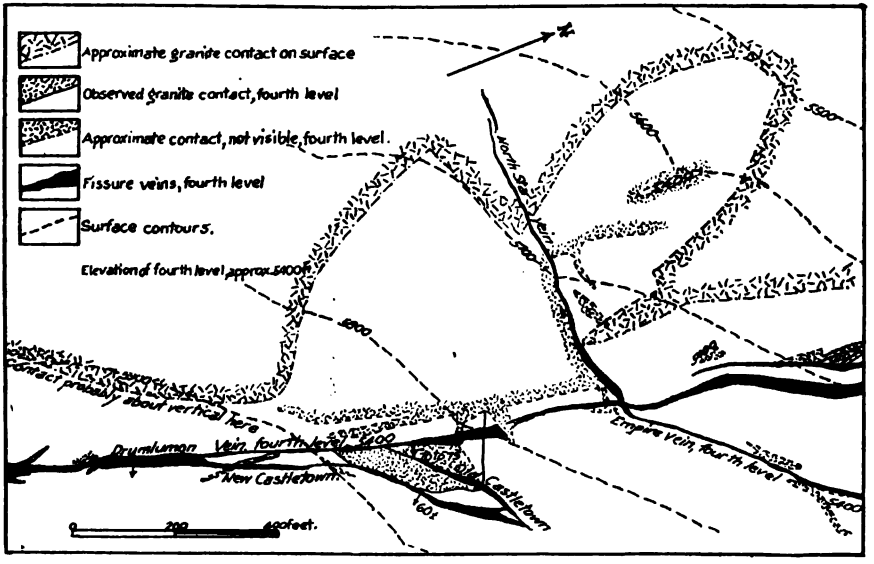


FIG. 2.—APPROXIMATE RELATIONS OF GRANITE BOUNDARY BETWEEN THE SURFACE AND THE FOURTH LEVEL, DRUMLUMMON MINE.

In *Bulletin* No. 213 of the U. S. Geological Survey, W. H. Weed gives a good description of the Drumlummon vein, from which the following quotation is made:

"It is a fault plane with white opaque quartz inclosing angular fragments of black, green and drab slates, which are sometimes distinct and unaltered and at others have been much decomposed. Where the ore-bodies are found the replacement has been complete and the former presence of the fragments is only recognizable by the outlines of the banded quartz. . . . . This vein, which is the largest and the most productive in the district, consists in its lower levels of a mass of angular rubbish, derived from the walls of the fissure, and in places cemented by quartz, in other places still retaining its original character. . . . . It may also be said that the ore shoots were well defined

and the intervening vein matter barren and unworkable. The pitch of the ore shoots conforms to the usual habit, dipping to the right when looking down the dip of the vein."

The walls are usually indistinct in the productive parts of the lode, but in the barren portions they are well defined. T. A. Rickard in his paper on Vein Walls[2] gives some interesting notes regarding the walls of the Drumlummon vein, and he also comments on the fact that the vein is only productive where it crosses the strata of the slates at an oblique angle, being unproductive where the crossing is at, or near, right angles.

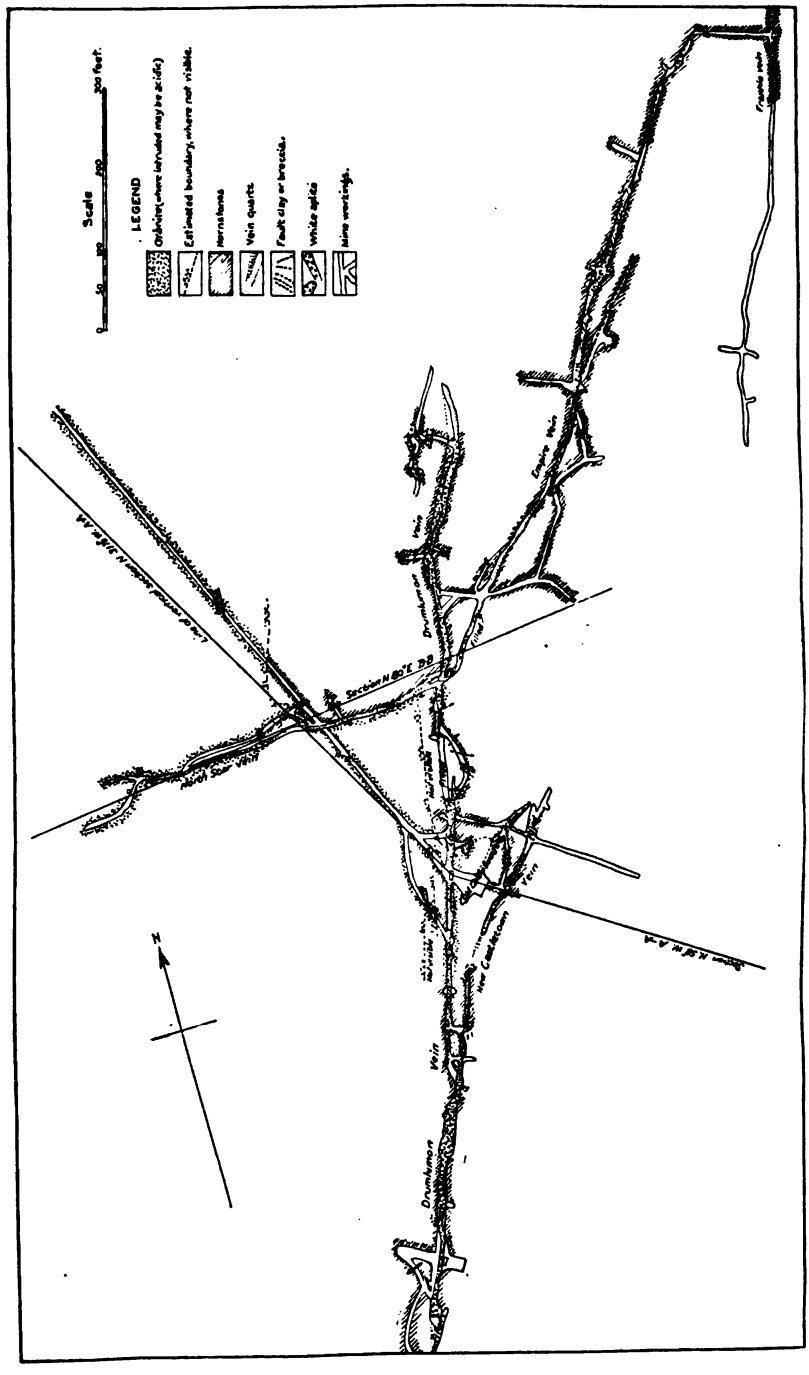
The Castletown lode has a strike about N. 40° E. and dips to the eastward at about the same angle as the Drumlummon. It joins this vein on the hanging-wall side in the "No. 1 South shoot." The walls are slate, or hornstone.

In 1893 an important discovery was made when, after a careful examination of the Castletown workings in and above the 2A level, it was decided to drive a crosscut into the hanging wall of that vein on the No. 3 level. At a distance of about 40 ft. a new orebody was found, which was soon opened up on all levels down to the 1,000, and yielded a large tonnage of good ore (Fig. 7). When fully developed it was found that this vein, which was given the name New Castletown, united with the Castletown on strike to the northeast, and formed a junction on its course southerly with the Drumlummon lode in the Sampson shoot (Fig. 2). The discovery of this new orebody led to further hanging-wall explorations, and one crosscut 210 ft. in length was run into the hanging wall of the New Castletown on the 400-ft. level, but without result.

The North Star vein, with a strike of N. 80° E. and nearly vertical dip, was very productive and yielded pay ore to a greater depth than the Drumlummon, or to the 1,200-ft. level (Fig. 8). It was given this name in the early history of the mine when it was supposed to be the same vein as the one discovered in the North Star claim, but later developments proved that the latter vein was the same as the Castletown.

The relation of the so-called North Star vein to the others of the system is shown in Fig. 3. In the professional paper by Joseph Barrell, previously mentioned, he says: "The other important vein is the North Star and its continuation, the Empire. This cuts across the Drumlummon and on the seventh level, where the relations are well exposed, it is seen to be younger than the Drumlummon, but has not faulted the older vein, which is here rather poorly developed, consisting of a mesh of quartz veinlets in brecciated slates." This statement is in conflict with views held by many geologists who have studied the mine; for at the intersection of the two veins on the 400-ft. level, there are evidences that the Drumlummon faulted the North Star, although it must be admitted that their

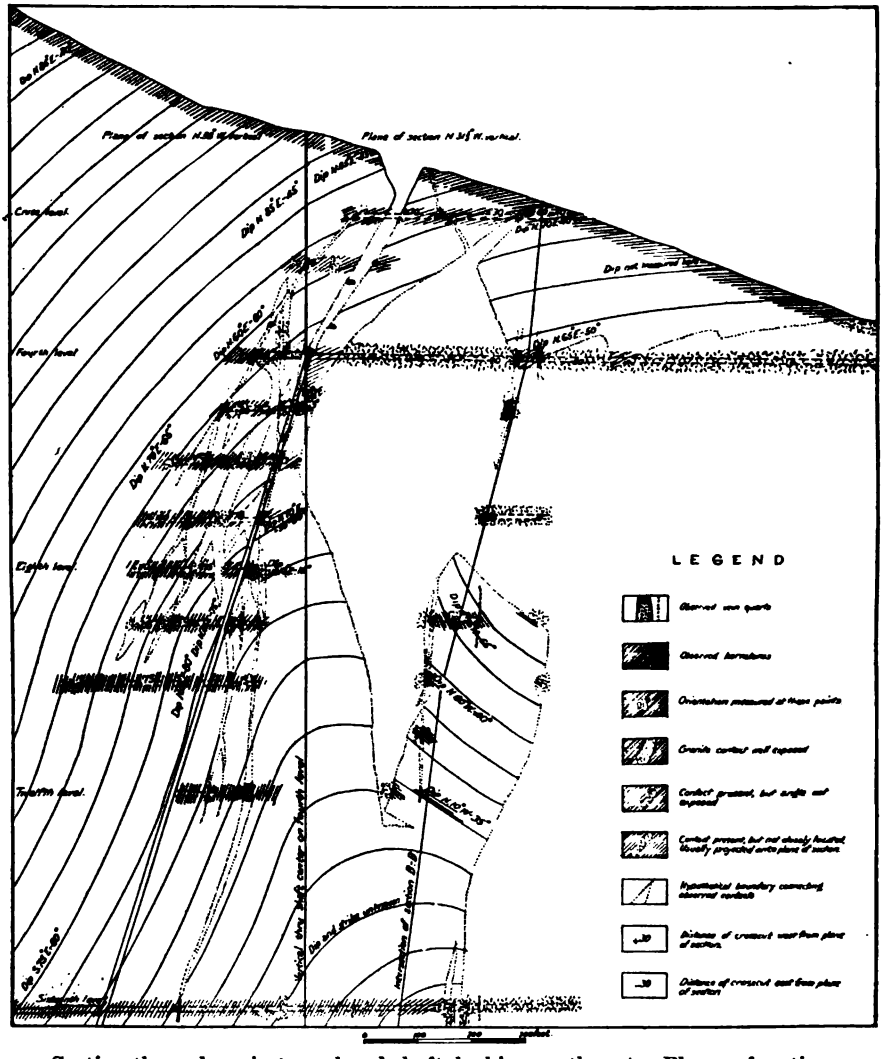
[2] *Trans.*, xxvi, 193 (1896).



Geology by R. W. Stone, 1901.

FIG. 3.—GEOLOGIC MAP OF FOURTH LEVEL, DRUMLUMMON MINE.

attention was never called to the condition on the 700-ft. level, which led to Mr. Barrell's conclusions. There is some foundation for the belief that the North Star does not extend beyond or into the hanging-wall of



Section through main tunnel and shaft, looking southwest. Planes of section, N. 58° W. and N. 31½° W.

FIG. 4.—VERTICAL SECTION, DRUMLUMMON MINE.

the Drumlummon, and that the Empire vein, like the Castletown, joins the Drumlummon on its hanging-wall side; the occurrence of the two veins in the walls of the Drumlummon at points nearly opposite each other being only incidental. Supporting this view are two facts: First that the strike of the Empire (N. 35° E.) is quite different from that of the North Star

(N. 80° E.), and is more inclined to be parallel with the Castletown. Second, mineralogically, the Empire resembles the Castletown and the Drumlummon, and is not at all like the North Star. The ore extracted from the latter contained very little silver, and the only sulphide mineral observed was pyrite, while from the former, high-grade silver minerals were obtained, and also some copper in combination with sulphur, antimony, and arsenic. In the oxidized portions a good grade of ore in the Drumlummon, Castletown, and Empire veins was indicated by the presence of carbonate of copper, and some fine specimens of azurite were found. Perhaps the difference above noted will sustain the contention that the veins were not of the same age, but if so, which is the older? If it is argued that the mineral character of a vein is influenced by the nature of the country rock in which it occurs, and not by its age, why was there no change mineralogically in the North Star when it passed out of diorite into hornstone or slate? The ratio of gold to silver in the North Star ore was about 1 to 3 in ounces, and this held true where the vein had slate walls, as well as where the inclosing rock was diorite, while in the other veins above mentioned the ratio varied from 1 : 10 to 1 : 20. It is worthy of note that nowhere along the contact between the diorite and hornstone were bodies of ore found.

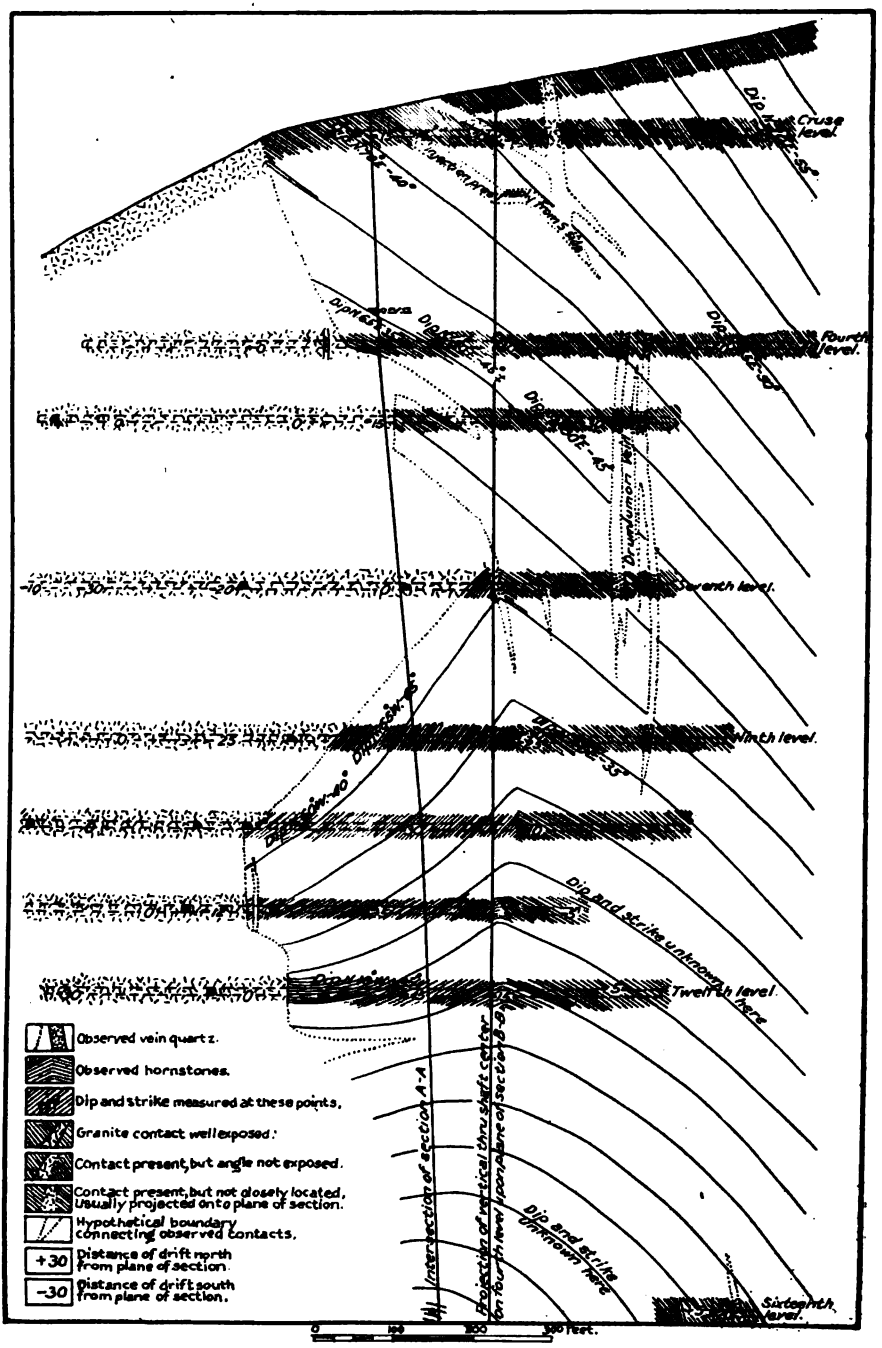
The discovery of pearceite, an arsenical polybasite, in the Drumlummon mine is interesting in this connection. In 1895, Dr. Richard Pearce was shown some beautiful crystals of this mineral which had been found in the 500-ft. stopes of the New Castletown vein, and sent a specimen to Prof. S. L. Penfield of Yale, who gave a full description to the Colorado Scientific Society.[3] The analysis of the specimen was as follows:

	Per cent.
Sulphur	17.71
Arsenic	7.39
Silver	55.17
Copper	18.11
Iron	1.05
Insoluble	0.42
	99.85

Professor Penfield says:

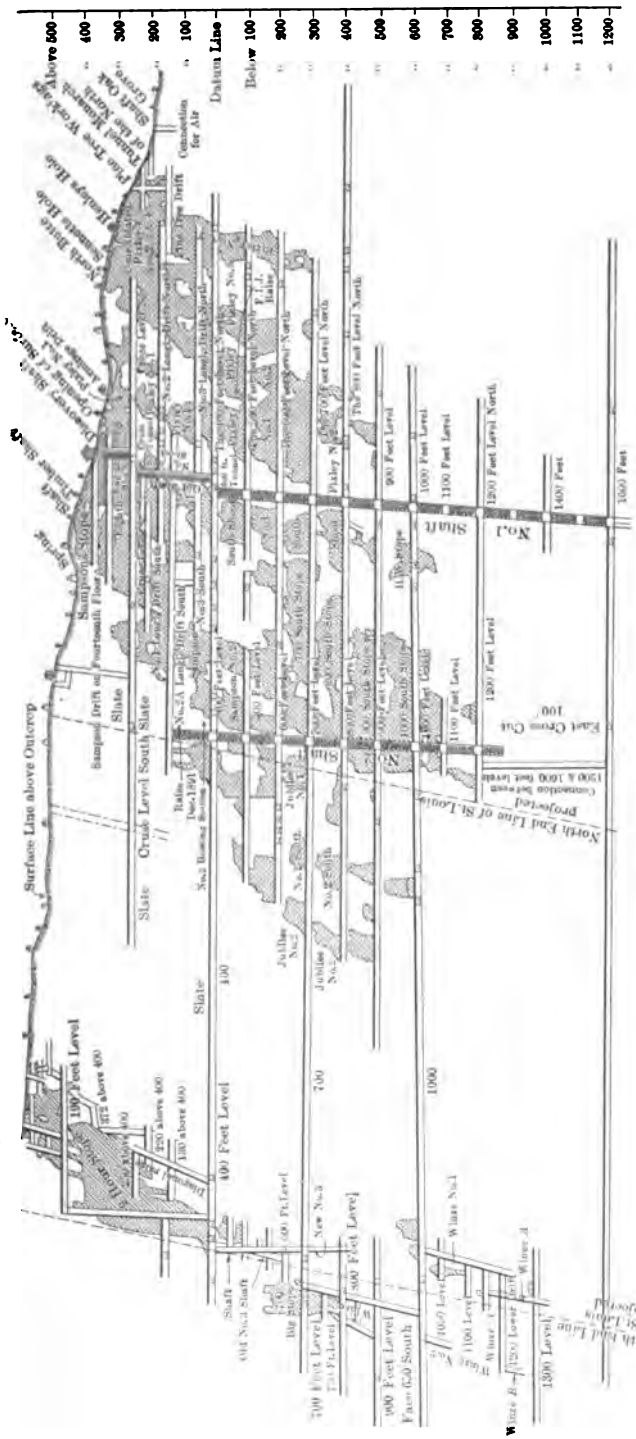
"It cannot claim to be a *new mineral*, for as an arsenical variety of polybasite it has previously been recognized, although no special name has been assigned to it. . . . . While recognizing that antimony and arsenic are isomorphous and may mutually replace one another, it is customary and has been found convenient in mineralogy to consider the sulphantimonites and sulpharsenites as distinct species, and to designate

[3] "On Pearceite, a Sulpharsenite of Silver, and on the Crystallization of Polybasite," *Proceedings Colorado Scientific Society*, vol. v, p. 210 (1894–1896).



Section parallel to North Star vein, looking north. Plane of section, N. 30° E.

FIG. 5.—CROSS-SECTION, DRUMLUMMON MINE.


Above 500
400
300
200
100
Datum Line
Below 100
200
300
400
500
600
700
800
900
1000
1100
1200
Surface Line above Outcrop
Crude Level South Slate
Slate
Sampson Slope
Sampson Drift on Fourteenth Floor
Connection for Air
Shaft
100 Feet Level
175 Above 400
250 above 400
130 above 400
400 Feet Level
700 Feet Level
900 Feet Level
Face 650 South
1100 Level
1300 Level
1000
700
Jubilee No.2
Jubilee No.3
New No.3
No.2 Shaft
No.3 Shaft
No.2 Winze
No.3 Winze
Winze No.1
Winze No.2
Winze No.3
Winze No.4
Winze No.5
Winze No.6
Winze No.7
Winze No.8
Winze No.9
Winze No.10
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Winze No.19
Winze No.20
Winze

them by different names, and the author proposes that hereafter the name polybasite shall be restricted to the antimony compound Ag_9SbS_6, and to make of the corresponding arsenic compound Ag_9AsS_6 a distinct species. For the arsenical mineral he takes pleasure in proposing the name *pearceite* as a compliment to his friend, Dr. Richard Pearce, of Denver, whose keen interest in mineralogy and connection with one of the large smelting and refining works of Colorado have made him known both to scientific men and to those interested in the development of the mining industries of the Rocky Mountain region."

Some beautiful crystals of quartz replacements of calcite were found in a large vug in the 900-ft. workings of the North Star vein, where the walls were slate. Photographs were taken of the best specimens before they were sent to the Smithsonian Institute, Washington (Fig. 9).

Mining

In developing the mine practically no timbering was required in drifts and crosscuts, as the hornstone was very hard and there was very little

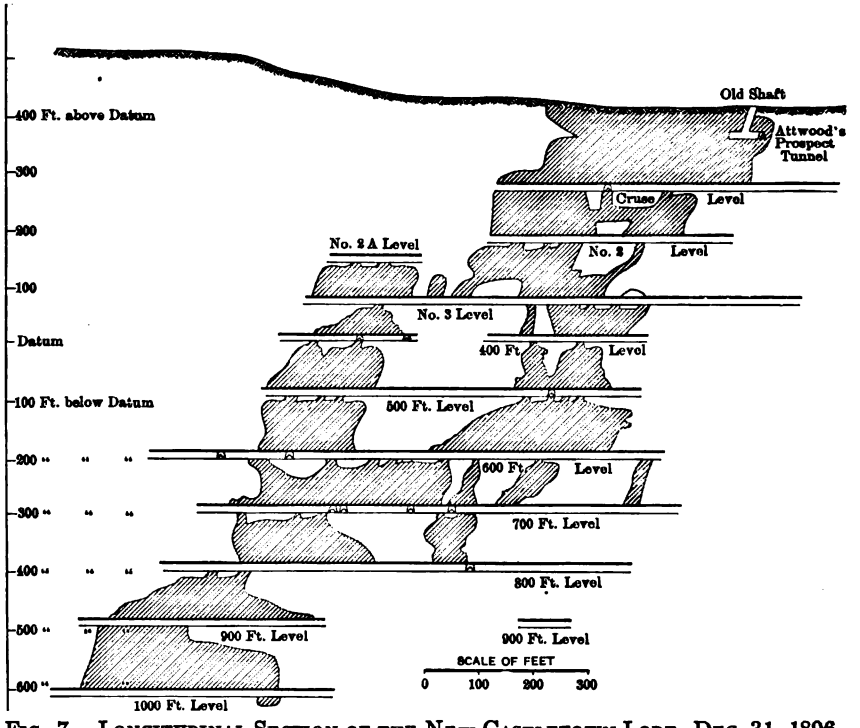


FIG. 7.—LONGITUDINAL SECTION OF THE NEW CASTLETOWN LODE, DEC. 31, 1896.

disintegration on exposure to the air. In the orebodies, after putting in stulls over the drifts and placing the chutes at proper intervals, stoping was continued without timbering, only enough ore being drawn out to

allow room for ore breaking, and therefore in the most productive era of the mine there was a large tonnage of broken ore in the stopes, the value of which was known from daily samples taken from the faces of the workings. Square sets were used in only one orebody in the mine, the 9-Hour, where the walls were treacherous, and the excavations had to be filled with waste as the stoping progressed.

In 1890, when new ground was being opened up rapidly in depth, it was apparent that provision should be made for a pumping plant of large capacity, and a Cornish pump was purchased, at an outlay of about $55,000, including partial installation at No. 2 shaft, where a large chamber was excavated and timbered in the hard hornstone. As the ground in deep levels drained down, the flow of water diminished greatly, and the installation of the Cornish pump was never completed, but in 1895 a Riedler, having a capacity of 400 gal. per minute, was placed in the 1,600 station of No. 1 shaft, and a great saving was made in draining costs, as this equipment replaced a system of pumps on the 700, 1,000, 1,200, and 1,600 ft. levels. The pump ran at only about one-third capacity at first, but extensive exploration in the 1,600-ft. level during the next two years increased the volume of water so greatly that it was considered advisable as a matter of safety to install another Riedler. This was a larger pump, its capacity being 500 gal., but it easily handled the entire flow, 468 gal. per minute under a head of 1,230 ft. Before this pump was installed, a crosscut had been driven westerly more than 700 ft. for the purpose of developing the North Star vein, and as the flow of water became somewhat alarming, a concrete dam was placed in the crosscut, so that the duty of the pump could be controlled. By the time the new Riedler was ready to run, the pressure at the dam went up to 264 lb. per square inch, representing a head of more than 600 ft.

The hoisting engine at No. 1 shaft was of the first-motion type, arranged to run single or in balance, and its cylinders were 20 in. in diameter by 60 in. long.

No. 1 and No. 2 shafts, in which cages were used, were of the same size, 14 by 4½ ft. in the clear, and were timbered with 10 by 10 in. Oregon fir, which cost $20 per thousand feet at the mine. The sets were 6 ft. apart, each set containing about 1,410 sq. ft. of timber, and the cost per foot for timbers was about $6.50, including labor of framing—the total cost averaging about $100 per foot. In explanation of this high cost it should be stated that it included the cost of excavating and timbering the stations, and that the rock was extremely hard and very difficult to break, owing to the angle at which the shafts crossed the stratification. Adolph Knopf calls the rock wollastonite and diopside,[4] but the analyses in Table I do not support this classification.

[4] Ore Deposits of the Helena Mining Region, Montana, *Bulletin No. 527, U. S. Geological Survey* (1913).

On May 8, 1892, a fire started in the 1,200 station of No. 1 shaft, and as the draft was strong through the level from No. 2 shaft, the fire raced up through the timbers with great rapidity. Before an adequate supply of water could be poured into the shaft, the flames had reached the 400 station, and 800 ft. of timbering was practically all destroyed. The fire was discovered by the night shift going into the mine. Hose lines were run into the tunnel from the fire pumps in the boiler room, and by stretching a wet blanket across the tunnel and pushing it ahead, the men were

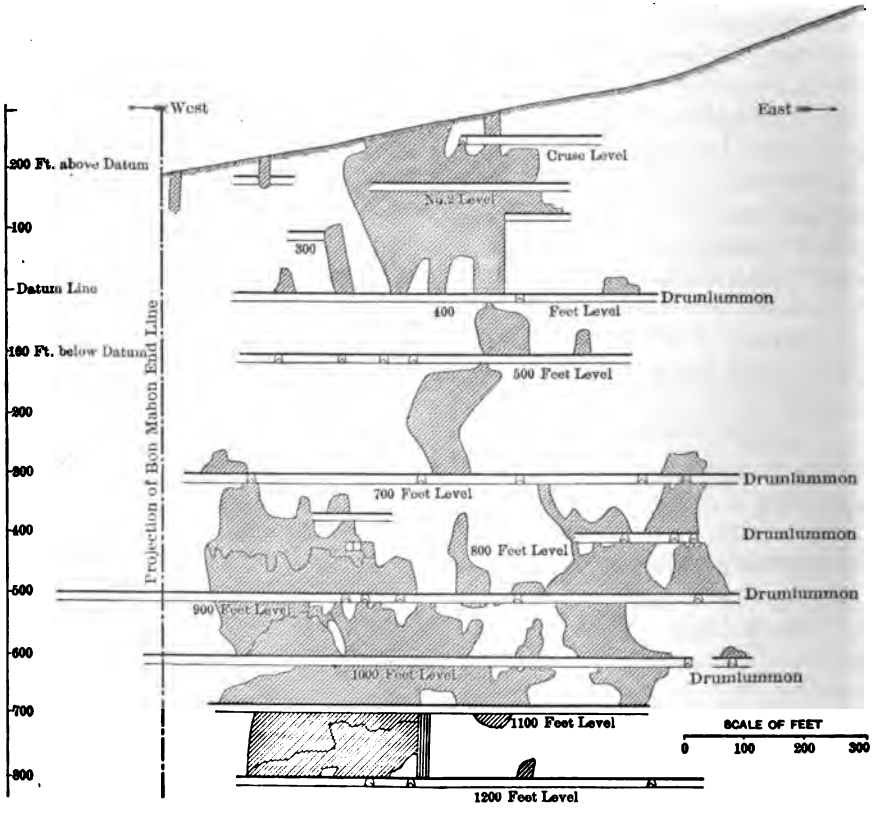


FIG. 8.—LONGITUDINAL SECTION OF THE NORTH STAR VEIN, DEC. 31, 1900.

able, several hours after the fire started, to get close enough to the shaft to put water into it, and also to save the station timbering and head frame. Not less than 5,000,000 gal. went into the shaft and the water rose to the 700-ft. level. The difficulty in extinguishing a fire in an incline shaft will be readily understood, for the water thrown into the shaft could not be effectively used. Nothing was done toward retimbering the shaft until the early part of 1893 and after the reorganization of the company. Owing to interruptions based on financial conditions, the repairs were not

completed to the 1,600 level until the middle of 1894. The burning timbers above the 1,200 had fallen down below that level, and the shaft was badly damaged all the way down to the 1,600 station. Furthermore, much of the timbering in the stations above the 1,200 had to be renewed, and the removal of the tangled and twisted pipes, which carried air, water, and steam, increased the cost of repairs; but as the shaft was in hard slate all the way, there were no serious caves. The origin of the fire was never known, but afterward while the superintendent was waiting in one of the stations for some blasts to go off, he noticed that the heavy

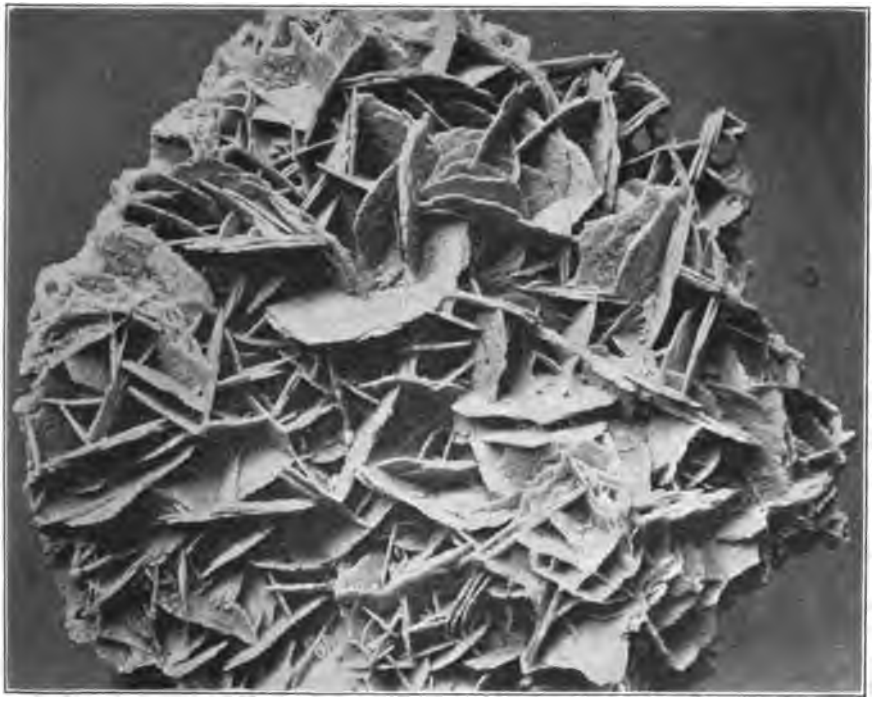


FIG. 9.—CRYSTALS OF QUARTZ REPLACING CALCITE.

concussions caused movement in a defective splice of the electric-light wires, and a fire started in some dry timbers just over the wires—so it was always regarded as quite possible that the fire in No. 1 shaft started in a similar manner.

The cost of drifting and crosscutting varied from $10 to $14 per foot, and the total mining cost per ton of ore was from $3.50 to $4, of which about $1.50 was for development work.

Milling

The first mill for the treatment of Drumlummon ore was erected by Thomas Cruse in 1878, and contained five stamps. Shortly after the

purchase of the property in 1883 by the English company, another battery of five stamps was added, and two amalgamating pans and a settler. In 1884 a 50-stamp mill was completed and provision was made for extracting the rich sulphides by treating the battery pulp on Frue vanners after it left the amalgamating plates. The feed boards of the vanners also carried amalgamating plates, and the mill was equipped with pans and settlers.

With the ore carrying about ½ oz. gold per ton, and from 7 to 12 oz. silver, the bullion from the plates was of about the following fineness:

	Gold	Silver
Battery plates	540	440
Vanner plates	380	600

There was a notable accumulation of amalgam on the plates.[5]

Two years later a 60-stamp mill was erected and began operations in November, 1886. It had two vanners for each battery of five stamps, but no pans and settlers, the duty of this mill having been to treat the lowest-grade ore from the mine, and particularly that low in silver. The stamps weighed 750 lb., the drop being from 7 to 9 in., 96 times per minute, the screens were of 30-mesh steel wire, and the average daily duty was 1.9 tons per stamp. The cost of this mill was $128,340.

In the years when the mine was making its best output the yield of the ore was about 1 ton of concentrate from 135 tons of ore, but as this product was very rich, carrying from $300 to $600 per ton in gold and silver, it contained about 22 per cent. of the values recovered from the ore.

The cost of treatment was about $4 per ton in the 50-stamp mill, and $1.15 in the 60-stamp mill.

Cyaniding

In 1896, under the direction of C. W. Merrill, tests were made in a small experimental plant, with a view to working the tailing from the mills by cyanide treatment, and the results were so favorable that a plant was built the next year, at a cost of about $66,000, and having a capacity of 400 tons per day. The equipment included several miles of railway of 36-in. gauge, with Porter locomotive and cars, the tailing having been impounded by five dams located along Silver creek for about 4 miles. Additions were made to this equipment a few years later,

[5] *Trans.*, xxvi, 33 (1896).

involving an expenditure of about $38,000, and by the close of 1907 all the tailing had been treated, amounting to 824,570 tons.

	Total	Per Ton
Value of product	$1,954,050.78	$2.37
Expenses	1,111,106.57	1.35
Profit	$842,944.21	1.02

	Tailing		Recovery		Residue	
	Total Contents	Per Ton	Total	Per Ton	Total	Per Ton
Gold, ounces	113,178	0.137	80,195	0.097	32,983	0.04
Silver, ounces	1,424,513	1.728	682,400	0.828	742,113	0.90

Percentage of recovery: gold 71; silver 48 per cent.

The average cost per ton was $1.35, but this included a charge of $0.125 for redemption of the capital invested in the plant. A summary of the costs per ton for the year 1904, when 95,490 tons were treated, follows:

Loading			$0.15
Transportation			0.10
Impounding			0.02
Supplies:	Cyanide	$0.375	
	Zinc	0.060	
	Lime	0.025	
	Fuel	0.115	
	Sundries	0.105	
			0.68
Operating labor			0.20
Superintendence			0.04
Total			$1.19

Composition of average precipitate:

	Per cent.
Gold	2.27
Silver	15.00
Copper	6.70
Zinc	38.00
Calcium carbonate	25.00
Insoluble	1.10

Production

The best year of the mine was 1887, when the yield of the ore averaged $27.21 per ton, and dividends amounting to nearly $920,000 were paid, but the grade of the ore did not keep up as depth was gained, and in 1892 a combination of misfortunes—the fire in No. 1 shaft, a flood, a bad cave in the 9-Hour stopes, and expensive litigation, made reconstruction of the company a necessity. This was effected Dec. 15, 1892, and a new name was adopted, the Montana Mining Co., Ltd.

The following table shows the production of the Montana Co., and of the Montana Mining Co., up to the time when the property passed to the St. Louis Mining & Milling Co.

	Ore Tons	Tailing Tons	Total Net Yield	Yield per Ton of Ore	Dividends
1883–1892 Montana Co.....	575,809		$8,498,794	$14.76	
1883–1892 Montana Co.....		14,925*a*	59,849		£537,057
1893–1910 Montana Mining Co.	572,641		4,719,131	8.24	
1893–1910 Montana Mining Co.		14,975*a*	34,000		
1893–1910 Montana Mining Co.		824,570*b*	1,954,051		90,355
	1,148,450		$15,265,825		£627,412

a Tailing treated in amalgamating pans.
b Tailing treated in cyanide plant.

Total Production	Oz.
Gold.....	568,898
Silver.....	4,982,942

The Penobscot and Belmont Mines

BY WALTER MCDERMOTT, LONDON, ENGLAND

Mr. Goodale has asked me for some notes on the other mines of the Marysville district, particularly the Penobscot, but it is more than 30 years since I was connected with that mine and the Belmont, and I am afraid I can contribute very little of interest. The following is mostly taken from old reports in my possession:

The Penobscot was located in 1872 by Nate Vestal, and created a great excitement a few years later from development of very rich ore near the surface. The first ore was worked in two small arrastres and a little five-stamp mill on Silver creek, a mile and a half below the mine. With this equipment $80,000 had been produced from ore yielding $20 a ton as an average.

In 1878 a 14-ft. vein of good ore was found. On the strength of this find the mine was purchased for $400,000 by the Penobscot & Snowdrift Consolidated Mining Co., the capitalization being $500,000. William Frue, inventor of the Frue vanner, with friends in New York, raised the capital for purchase and equipment, and controlled the operations for some time; he also purchased the Belmont mine.

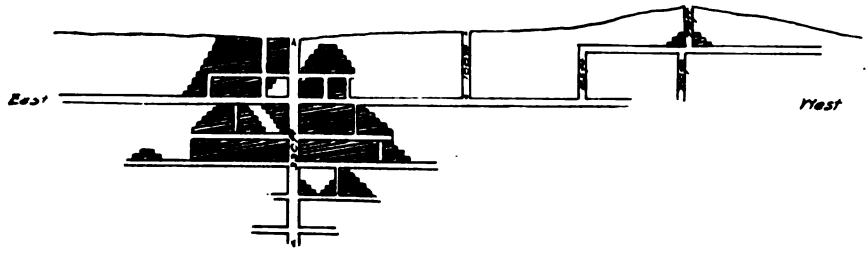


FIG. 10.—LONGITUDINAL SECTION OF PENOBSCOT MINE, MAR. 15, 1880.

The first ore taken from the big vein was very rich, 685 tons yielding $80,798 in the first five months of 1878. Mining and milling costs were about $10 a ton. This ore was treated in the arrastre, and the tailings from it assayed 1 oz. in gold and 2.05 oz. in silver. Some samples assayed from vein widths of 6 ft. yielded better than $1,000 gold per ton. The average silver content was about 10 oz.

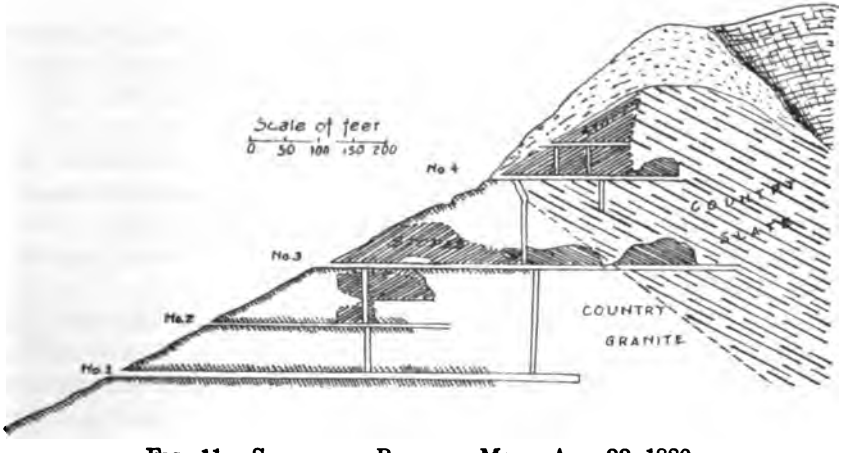


FIG. 11.—SECTION OF BELMONT MINE, APR. 22, 1880.

Reports made at the time the property was purchased were very flattering. The average assays of the workings were given as $30 gold and 3 oz. silver per ton.

In spite of this encouraging start, the results proved disappointing to the stockholders. The value did not persist in depth and the ore shoots

proved to be very erratic and pockety. The Bonanza shaft was sunk to a depth of 330 ft. but yielded unsatisfactorily below a depth of 160 ft. In addition to erratic values, the vein widths also varied rapidly. The vein was in silicified shale near a granite contact. Its strike was northeast-southwest and the dip was 75° to 80°.

During the period from September, 1878 to January, 1880, 14,307 tons of ore were mined and milled, yielding $251,661 in bullion. The expenses for that period, outside of purchase of mine, were $268,513, showing a loss of $15,852. From the above, the average bullion yield per ton is seen to have been $17.59.

Soon after this the property was closed down, remaining idle for several years. Later it was worked on tribute for a time.

Operating costs at that time were not so excessive as one might think, considering the cost of transportation and supplies. It was 204 miles from Helena to the terminus of the Utah & Northern Railway, 52 hr. by stage. The average mining costs, including extensive prospecting, were $8.60 per ton, and milling costs were $2.98 per ton.

The mine water, about 35 gal. per minute, was used in the mill. The 35-h.p. hoisting engine was also used to run the Blake crusher.

The ore was hoisted in iron buckets of 750 lb. capacity, sliding on skids in the inclined shaft. With a mine yield of 1,100 tons of ore per month, hoisting and pumping cost 68c. per ton.

A system of tram lines brought the ore from mine to mill. From the shaft the ore was trammed 200 ft. to the crusher, from which it was carried on a double-track gravity tram 3,300 ft. long to the ore house, and from there it was trammed 300 ft. to the mill. The gravity tram had cars of 2½ tons capacity, and could keep the mill going by running 2 hr. out of the 24. Tramming costs were figured at 29c. per ton.

The mill was considered a very fine one for those times. It was furnished by the Fraser & Chalmers Co. of Chicago, and had 40 stamps, each weighing 650 lb. The stamps dropped 70 times per minute with a 7-in. drop. Forty-mesh screens were used and the capacity was rated at 1½ tons per stamp per 24 hr.

Below the plates there were four Frue vanners, and the vanner tailings were run through blanketed sluice boxes. About 75 tons of concentrates, containing lead as sulphide and carbonate, and averaging $50 per ton, had accumulated up to 1880, but there was no smelter near enough to handle this product profitably.

A 68 per cent. saving was made by the mill, the tailings assaying $8.30 per ton.

The 40-stamp mill cost about $40,000. After the 40-stamp mill was in operation, the arrastre was no longer used. The arrastre plant was a rather pretentious affair, consisting of four grinding tubs, three amalgamating tubs, and one settler.

In 1880 the Belmont mine had a 20-stamp mill. The property looked very encouraging at this time, having yielded $104,238 from December, 1878, to April, 1880. The yield per ton was between $8 and $20. Mine and mill costs were about $6.50 per ton.

The Belmont vein was in silicified shale in the upper levels, but ran into the granite with depth. The ore paid fairly well in the upper levels, but petered out in value and regularity below. The change of country rock in depth appeared unfavorable in the case of the Belmont.

The Belmont mill had shaking copper plates in addition to stationary plates. These were attached to the Frue vanners as feed aprons, and, fitted with two low riffles (top and bottom), not only made an additional regular saving of gold after the fixed plates, but were effective in catching hard amalgam when careless work at the batteries in feed of quicksilver caused occasional losses. Four vanning machines concentrated the tailings from the plates. The concentrates were treated in an amalgamation pan and settler.

In general it may be said that all the early day mines of the Marysville district, except the Drumlummon, were disappointing in depth.

THE EMPIRE MINE

BY F. L. SIZER, DOS CABEZAS, ARIZ.

The Empire Mining Co., Ltd., of London, England, was organized early in 1886, to take over the Empire mine, owned by Hickey and Cotter, and the Whipperwill mine, owned by Nate Vestal, who had discovered the Penobscot mine a few years earlier and sold it for a half million dollars. The claims in the Empire group are shown in Fig. 12.

The Empire mine was little more than a prospect, opened by cuts and shallow shafts, none of them over 70 ft. deep, but already showing a good body of high-grade gold-bearing quartz.

The Whipperwill mine had been worked to a depth of 350 ft., and had produced largely, but, at the time of the organization of the Empire Mining Co., had not been in operation for nearly 10 years, and most of the workings were caved and inaccessible.

The writer became the first manager for the new English company and, after a few necessary repairs to the old 10-stamp mill, in Coombes gulch, a mile below the mine, started milling operations on ore which averaged about $20 per ton. This ore was readily amalgamated and a high percentage was usually saved. Inside amalgamation was the method in use in the old mill.

The locality on the west side of Belmont mountain, 5 miles from Marysville and 1½ miles south of Gloster, was a heavily timbered country, cut by deep gulches and subject to a heavy snowfall, which made the spring months difficult ones in which to start an operation.

Repair of old roads and building of new ones was the first order of business, and, while Marysville afforded some supplies, Helena, 27 miles distant, was the principal supply point, everything then coming by wagon over a heavy grade from Marysville, or the longer route, by Silver City and the Little Prickly Pear creek, a distance of 35 miles.

A telephone line to Marysville was one of the first things constructed and by midsummer of 1886 Empire was a bustling camp.

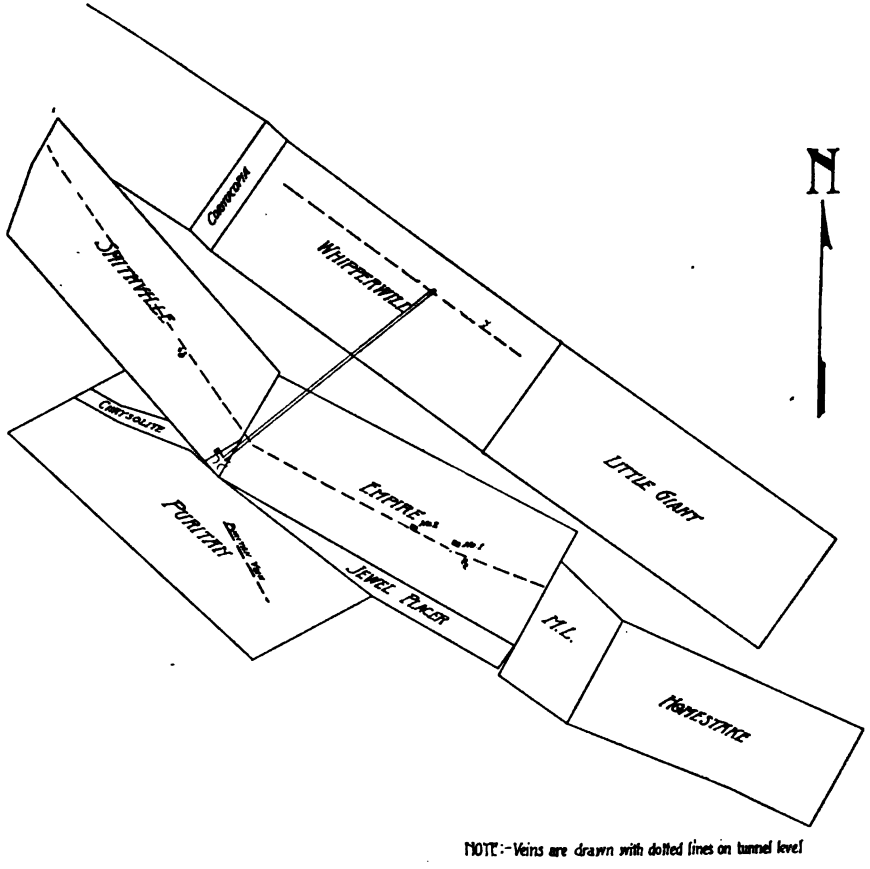


FIG. 12.—EMPIRE GROUP, STEMPLE MINING DISTRICT, MONTANA.

A daily stage to Marysville brought mail and passengers, and as much freight as a four-horse team could handle, sometimes obliging the passengers to walk up the steep grade. Occasional visitors who drove out by team from Helena thought the descent into Coombes gulch altogether too steep for wheeled vehicles and sometimes tied their teams at the top of the hill, coming down to the mill on foot.

The Drumlummon mine was in full swing, with the 50-stamp mill in operation and the 60-stamp gold mill under construction. A number of the same Englishmen were large owners in both properties.

The initial development of the Empire mine was the driving of an adit at the west end of the claim, through the hard metamorphosed black slate which, at a distance of 120 ft., cut the Empire vein. From this point levels were driven, both east and west on the vein, and this constituted the main working level.

I well remember that this rock was so hard that a good crew of miners who had taken a contract for driving the adit at $20 per foot, threw it up in despair at less than 40 ft. in. This adit was afterward extended straight into the mountain, in a northerly direction, and at 1,100 ft. cut the Whipperwill vein at a depth of 500 ft. where the vein was practically barren. The entire length of this 1,100-ft. working tunnel was in the metamorphosed black slate, without any intrusion of quartz-diorite or porphyry and, so far as my development of the Empire vein went, there was no change in the country rock.

The higher-grade Empire quartz was generally slightly copper stained, and all the surface ore highly oxidized, iron predominating. There was enough copper in the tailings to prove a serious drawback to their treatment by the cyanide process, which was attempted several years later, but not enough for us to receive pay for the copper in the concentrates, except in one shipment. There was very little silver in the ore, generally an ounce of silver to the ounce of gold, and the bullion ran about 700 in gold fineness, 298 in silver and 2 in copper. The Whipperwill quartz contained no copper, was generally harder, and the better grade of ore was the rose quartz and purple-stained amethystine quartz, some of the crystals obtained there being beautiful cabinet specimens.

The Empire vein had a nearly east-and-west strike, dipping to the south at 70°, while the Whipperwill, parallel to the Empire vein in strike and occupying the crest of the ridge, was dipping north at about 80°. There were no cross veins intersecting the Empire and Whipperwill veins and no faulting of either of them, although the Smithville (the western extension of the Empire vein) had a strike of about N. 60° W., and there was a radical change in the character of the vein quartz on this west extension of the Empire.

The richer portion of the Empire was between the surface and the 300-ft. level (the adit above mentioned being the 420-ft. level), and on the 300-ft. level, and just above it, was some of the richest gold-bearing quartz ever taken out of any mine in the district.

A peculiar feature here was a slab of galena ore on the hanging-wall, very rich in gold and associated with copper-stained quartz, which extended for a length of 150 ft., a height of 40 or 50 ft., and a thickness of from 2 or 3 in., on the edges, to a thickness of 10 in. in the middle. At

no other point in the mine was galena ore ever encountered. The Empire vein had an average width of 6 ft., some of the stopes opening to a width of 16 ft.

Below the adit mentioned a winze was sunk to a depth of 130 ft., and levels run from the bottom, east and west, making this depth from surface 550 ft. below the highest point of outcrop at the east end of claim. At this depth there was no pay ore, the vein showing for the most part as a fissure filled with crushed slate, devoid of value, and what quartz there was tending to sprangle out into the slate walls as insignificant seams. No pay ore was taken out below the tunnel level, with the exception of about 40 ft. in depth, and this was a short shoot.

The Whipperwill vein was about 4 ft. in width, opening to 10 ft., but at a depth of 500 ft. was so uniformly low grade as to preclude the possibility of working it at a profit. This was also true of the Smithville, the western extension of the Empire vein, which was devoid of copper-stained quartz and was generally of a value below $6. There was no exploration of the Whipperwill vein below the working-tunnel level, during my time, though I have some recollection of having heard that a shallow winze was sunk, in later years, with no profitable results.

During 1886, amalgamation, both inside and outside the mortars, was tried and a five-stamp battery of light stamps, high drop, for an experiment, and Frue vanners for saving the concentrates, were added to the old mill. With the high-grade surface ore a fairly prosperous year resulted; so that before the end of the first year's existence of the Empire company a 15 per cent. dividend was paid, and authorization made for a new mill.

The concentrates did not constitute more than 5 per cent. of the total mine output; and were usually below $100 per ton in value, and from only one locality, that above the 300-ft. level, as mentioned, did I ever receive pay from the smelters for lead in the concentrates, although all the concentrates carried a considerable percentage of lead, much of it in the form of carbonate. One shipment gave the following assay: Gold, 4.2 oz., silver, 29.4 oz.; lead, 36.5 per cent. The total output in two years, under my management, was a little over a half-million dollars—all from the Empire and Smithville veins, but none from the Whipperwill.

Doubtless because of the fact that the Montana Co., Ltd., at Marysville, had just completed a 60-stamp mill, the Directors declared in favor of a mill of the same size at Empire, and by April of 1887 the work of excavating for foundations was begun. Deliveries of lumber and machinery were slow, owing to the long wagon haul, so that it was late in November before the first 40 stamps were in operation and the remaining 20 stamps, about the end of January, 1888. A substantial and up-to-date mill was the result, and a monthly crushing capacity of 5,000 to 6,000 tons was the average.

To supply this larger mill very rapid work in the mine was necessary, and, because of the failure of the lowest levels to develop any new ore, a lower average grade ore was supplied to the mill. A rail tramway connecting the main adit level with the mill proved a cheap method of delivering the ore, and the whole operation was systematized and constituted a model operation, stimulating other prospecting and mining work in the district.

Marysville became a lively center and, for the succeeding eight or ten years, was a prosperous mining camp, obtaining rail connection late in the year 1887.

The Stemple district has always been one prolific of profitable gold-quartz mines and now, after the lapse of 25 years, has again taken on new life due to renewed activity in the Bald Butte, Piegan-Gloster, and others of the old mines, re-opened, and it is quite likely that further prospecting will uncover some new bonanza orebodies in the mines of this district.

The Empire mine is one of those which has been operated at intervals since 1893, by a number of different owners, but has been idle for a major portion of the time in the past 20 years. At one time it was reported that a payable body of copper ore had been discovered in the lowest level, but it proved to be only a limited amount of the heavily copper-stained quartz, and even that not specially high-grade gold ore.

It has many times been suggested to me that the lowest level in the Empire was in a barren zone, and that deeper sinking would result in penetrating the continuation of the ore shoots, which were so rich and profitable above.

If any operation in the district is continued long enough to test this surmise, it will be a distinct gain for the district, as a whole, and I hope that some purse will be long enough to give this matter a fair trial.

I confess that the very hard and highly metamorphosed slates at the head of Coombes gulch and general underground conditions in the Empire mine, with many quartz veins which, at shallow depths, 200 ft. or less from the surface, die out in the hard slates, suggests the thought that some of the quartz veins higher up on Belmont mountain than the Empire are likely to be more lucrative to re-operate than this mine, which was my first love in the grand old State of Montana.

Regretting that my scattered notes of earlier years of experience do not permit of my giving a more detailed account of this very interesting property, and hoping that these brief notes will stimulate some of my contemporary engineering friends in Montana to add to this line of historical record, I herewith submit my contribution.

Copper Ores of the New London Mine*

BY B. S. BUTLER AND H. D. McCASKEY, WASHINGTON, D. C.

(Salt Lake Meeting, August, 1914)

Introduction

THE New London copper mine, about 8½ miles east of Frederick, Md., was visited by the writers for a few hours in the spring of 1909 and the following brief notes on ore specimens collected are presented as a contribution to studies in chalcocite ores.

For certain data on the structural and areal geology of this portion of Maryland the writers are under obligation to the unpublished notes and maps of Arthur Keith. The detailed study of the ores is the work of B. S. Butler.

The deposit here discussed has many similarities to those of the Virginia district of Virginia and North Carolina, which have been described by Graton[1] and Laney,[2] except that in the deposits of the New London mine chalcocite is by far the most important ore mineral, while in the Virginia deposits bornite is commonly the most important sulphide. In the New London occurrence calcite is the most important gangue mineral.

Field Relations

The town of Frederick lies in the center of a narrow belt of limestone ranging in age from Ordovician on the east to Cambrian on the west. This belt is about ½ mile wide at Noland's Ferry, on the Potomac, and broadens to a width of 3 miles at Frederick. The general direction is east of north. The eastern boundary is, in places, a fault plane along which a series of metamorphic rocks of presumptive pre-Cambrian age have been thrust up. These ancient rocks, for several miles east and west of New London, are phyllites in the main, but closely interfolded near the New London mine are long, narrow lenses of epidote

* Published by permission of the U. S. Geological Survey.

[1] Graton, L. C.: *Mineral Resources of the United States*, pt. 1, p. 620 (1907).

[2] Laney, F. B.: *Economic Geology*, vol. vi, No. 4, pp. 399 to 411 (June, 1911).

schist, the "greenstone" of the region, and slender lenses of marble and of quartzite, the entire series lying in the characteristic northeasterly strike of the Appalachian rocks. Crossing the boundary fault at a very slight angle is a persistent narrow diabase dike, probably of Triassic age, many miles long—the filling of a deep fissure cutting both Ordovician and pre-Cambrian rocks. The New London mine lies nearly in the center of the elongated group of exposures of epidote schist, which in turn lie about in the center of the broad belt of phyllites. The phyllites in general strike northeasterly and dip at high angles southeasterly. They are fine-grained, dark, and crumpled to microscopic degree. Their origin, whether sedimentary or igneous, is not wholly determined, though part are pyroclastic and part are sedimentary. The epidote schists are in part amygdaloidal—the characteristic Catoctin schists or greenstones of the Appalachians—and they are derived by regional metamorphism of ancient basaltic extrusives. The marble and the quartzite are considered to be of pre-Cambrian age. Intense forces have crumpled all these formations, and their present structure, where not massive, is trough or saddle form.

About ½ mile west of the New London mine a narrow oxidized manganese vein is found at the contact between epidote schist above and marble below, dipping 55° S. 80° E. and striking N. 10° E. The strike of small pits and shafts along the outcrop of the New London vein was found to be about N. 62° W., magnetic, indicating a fracture across the prevailing strike of the rocks. The succession of rocks at the mine is marble overlain by epidote schist, in turn overlain by quartzite. Below the marble lens, and above the quartzite also, lies the soft, crumpled, fine-grained, black phyllite. The ore appears confined mainly to the marble, and ore solutions were probably limited in circulation by the phyllite. It is also assumed that the resistant marble and quartzite afforded the necessary fracturing for passage of the ore solutions, as the surrounding soft phyllites are tight, finely folded rocks and do not suggest openings of any kind.

The New London Mine

The New London mine is an old property on the southern end of the western copper belt of Maryland which extends northeasterly to the Liberty mine, formerly the largest copper producer of the State. The total output of copper of all Maryland mines has been relatively small, but these deposits have furnished limited quantities of rich ore. The principal workings of the New London mine in 1909, at the time of our visit, were a shaft 210 ft. deep and a stope in the bottom. Older workings consisted of an inclined shaft to the 173-ft. level and a number of stopes, all of which were pretty well filled with waste. The ore from the bottom

stope was principally chalcocite, with minor quantities of bornite, and with gangue of coarse crystalline pinkish calcite, blebs of white quartz, and interfolded and crushed country rock. The small output has consisted in part of hand-sorted ore running 10 to 11 per cent. in copper and more recently of concentrates of higher grade. Sampling of the ore at a depth of 210 ft. is reported to have shown an average content of 4.9 per cent. of copper. Pyrite was reported in small quantity at the bottom of the 200-ft. level, but none was observed by the writers.

Since the visit of 1909 a 100-ton concentrating mill has been put up on the property and the new inclined shaft has been sunk to the 300-ft. level. On the 200-ft. level the drift has been extended 305 ft. easterly and 95 ft. to the west. The ore shoot is reported to extend 150 ft. on this level, with an average content of 3.6 per cent. copper.

Description of the Ores

The ores may be roughly divided into two types, that may be designated as (1) typical "vein" ore and (2) banded or schistose ore.

The first type, so far as observed, occurs in the wider portions of the vein. It consists of coarsely crystalline calcite, quartz, and sulphide. Barite was noted in some of the thin sections examined, but was not detected in the hand specimens. Chalcopyrite[3] is reported from the district, but was not observed by the writers. The sulphide is mainly chalcocite, with bornite scattered in small particles through it. The calcite and quartz are in crystals and aggregates, in some cases exceeding an inch in diameter.

The calcite varies from nearly white to light pink in color. No quantitative chemical work has been done on this mineral, but it contains the merest trace of magnesium, manganese, and iron. The presence of manganese is shown by the oxides of this metal that result from the surface alteration. The quartz and calcite are contemporaneous in origin, each mineral including the other, and the barite observed is contemporaneous with the carbonate, though its relation to the quartz was not noted. A large part of the sulphide in this type of ore is later than the quartz and calcite. It surrounds the crystals and enters along fracture and cleavage planes, and replaces both quartz and calcite. (See Fig. 1.) To a slight extent the sulphide is contemporaneous with the gangue minerals, being completely inclosed in quartz and calcite crystals that have surrounded the sulphide grains in the process of growth. The chalcocite is notably well crystallized, crystal faces exceeding $\frac{1}{4}$ in. in greatest dimension being present in the specimens.

Bornite occurs in small irregular grains through the chalcocite. At first glance the sulphide appears to be pure chalcocite, but on close ex-

[3] Weed, W. H.: *The Copper Mines of the World*, p. 267 (1907).

amination small irregular areas of bornite can be seen in all the specimens of massive chalcocite examined.

Microscopic examination of polished and etched faces of the sulphide shows that very commonly the bornite occurs between the chalcocite crystals and frequently occupies fractures in them. (See Fig. 2.) In other instances there is a parallel intergrowth of lath-like crystals of bornite and chalcocite. In such cases the several lath-like bodies are apparently portions of an individual crystal, all having a uniform orientation. In many instances the bornite occurs as small specks through the

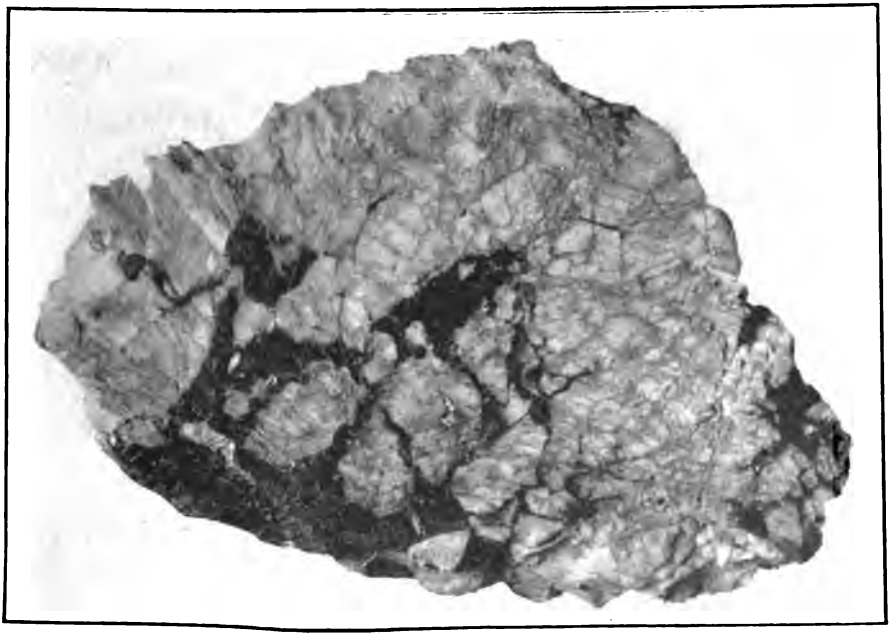


FIG. 1.—"VEIN" ORE, NEW LONDON MINE, SHOWING RELATION OF SULPHIDE TO GANGUE MINERALS.
Dark areas, Sulphide. Light areas, mainly Calcite.

chalcocite, but frequently with a definite arrangement that suggests intergrowth.

A similar relation exists between different chalcocite crystals as that between chalcocite and bornite. Lath-like crystals of chalcocite are frequently inclosed in irregular chalcocite crystals, and chalcocite crystals occur between other crystals and extend into fractures.

As already indicated, some of the bornite is undoubtedly later than some of the chalcocite, and in no instance is there positive evidence that the bornite has been replaced by chalcocite. For the most part the two sulphides are contemporaneous. The interlocking of the chalcocite crystals is similar to that resulting from the crystallization of a metal

or that of a granular igneous rock. There is no structural indication that the chalcocite has resulted from the replacement of an earlier sulphide.

The banded or schistose ore may occupy the entire "vein" in the narrower parts or only a portion where the "vein" is wider. Typically it exhibits a finely laminated structure with successive layers of light and dark bands. In some specimens the individual layers have a thick-

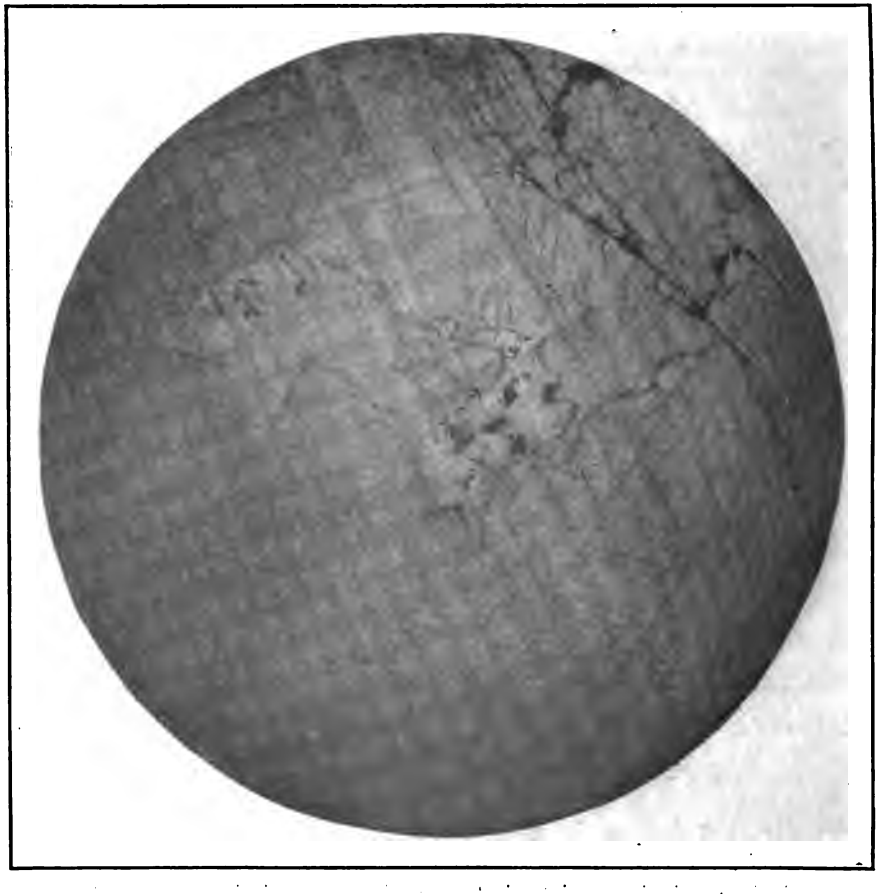


FIG. 2.—"VEIN" ORE. × 30. SHOWING RELATION OF CHALCOCITE AND BORNITE. Small bodies lying between larger crystals and connected by narrow stringer, Bornite. Large crystals, Chalcocite. Bornite occupies fissure in Chalcocite.

ness not exceeding $\frac{1}{16}$ in., while in other instances bands composed largely of dark or light minerals will have a thickness of $\frac{1}{2}$ in. These banded ores include lenticular nodules in which the banding is much less distinct. The banded ore has the general appearance of a schist that has resulted from the metamorphism of a rock in which some of the more

resistant portions still retain their identity in part though drawn out into lenticular form. (See Fig. 3.)

In this type of ore the light bands are composed mainly of calcite and quartz, calcite being much more abundant than quartz, while the dark bands are due to the presence of the copper sulphides. Near the walls of the "vein" and in some instances at some distance from the

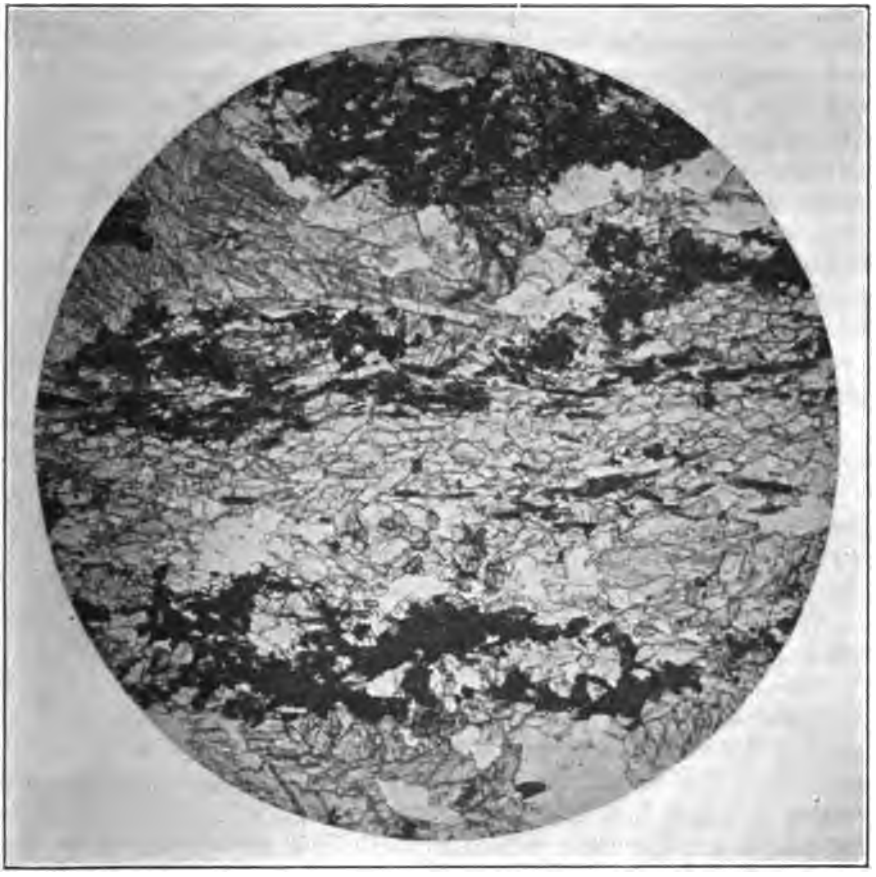


FIG. 3.—"SCHISTOSE" ORE. × 30. SHOWING RELATIONS OF SULPHIDE TO GANGUE MINERALS.
Dark areas, Sulphide. Light areas, mainly Calcite.

walls, chlorite, specularite, and muscovite are abundant. Tourmaline and titanite are locally abundant in the banded ores and feldspar is present in small amounts. The calcite, quartz, and sulphide of this type of ore have been crushed and recrystallized. Abundant examples are to be seen where each of these minerals includes the others, indi-

cating contemporaneous crystallization. The brittle sulphide has been crushed and drawn out into bands composed of small fragments.

The tourmaline occurs in small crystals showing pleochroism from light to dark brown. These can rarely be seen with the unaided eye, but are readily detected with the hand lens when their presence is known. Under the microscope they are found included for the most part in the other minerals, but rarely sulphide is inclosed in the tourmaline crystals. The crystals in general lie in a plane parallel to the banding. (See Fig. 3.) The mineral appears to have formed in the early stages of the recrystallization including fragments of the sulphide, but later it was included in the carbonate, quartz, and orthoclase, and also in recrystallized chalcocite. The titanite is in small granules apparently formed at about the same time as the tourmaline and the two are frequently intimately associated.

As already stated, near the walls and apparently wherever the wall rock has been included in the vein, chlorite and specularite become abundant in the ore and muscovite is of common occurrence. These minerals are similar in their character and occurrence to those in the schists which form the wall rock.

The sulphides recognized in the schistose ore are chalcocite, bornite, and pyrite.

The relations of the chalcocite and bornite are in every way similar to those already described. Only a few small grains of pyrite were noted. They were apparently among the first minerals formed, and in no instance observed was there any replacement of pyrite by copper sulphide.

Relation of the Two Types of Ore

Where the two types of ore come together there is commonly a gradual transition from the one to the other.

This relation suggests that the schistose ore may have resulted from a metamorphism of the vein due to slight movement under conditions of considerable heat and pressure. The dynamic metamorphism of ore deposits in the Appalachian region has been described by Emmons,[4] and need not be discussed here further than to say that it has been noted by several observers in widely separated localities in the southern as well as the northern Appalachians.

Genesis of the Sulphides

The genesis of the deposits will not be discussed except as to the origin of the sulphides; namely, whether they are "secondary"—that is, resulting from the replacement of earlier sulphides—or whether they were deposited essentially as they are now except as affected by dynamic metamorphism.

[4] Emmons, W. H.: *Economic Geology*, vol. iv, No. 8, pp. 755 to 781 (Dec., 1909).

It has already been stated that there is no evidence of the chalcocite having replaced either bornite or pyrite, the only other sulphides recognized in the deposit, and further, that there is no indication in the structure of the chalcocite that it has replaced any earlier sulphide.

The intimate intergrowth of the sulphides with carbonate seems a strong additional argument against considering them as resulting from downward sulphate solutions. Theoretically one would expect the copper sulphate solutions would react with the carbonate of the gangue to form the relatively stable copper carbonate and that there would be slight migration of the copper solution. Such is known to be the case in instances where sulphides are intimately associated with a carbonate gangue.[5] Spencer[6] has recently shown that calcite does not precipitate copper from a solution which contains both cupric and ferrous sulphates. Some of the carbonate in the gangue of the ore contains a little iron, but that it is a precipitant of copper is amply shown by the presence of malachite along cleavages in the carbonate where incipient oxidation of the chalcocite has taken place. It has also been shown experimentally that the calcite of the gangue will readily precipitate copper as carbonate from a sulphate solution. It does not seem probable, therefore, that the copper could have been transported as sulphate and have replaced another sulphide entirely inclosed in a carbonate grain.

The coarse crystalline structure of the "vein" ore and gangue suggest deep-seated and not surface origin.

The evidence seems to the writers to point to the conclusion that the chalcocite, which is the principal sulphide in the deposit at the New London mine, was deposited without the replacement of an earlier sulphide.

[5] Bard, D. C.: *Economic Geology*, vol. v, No. 1, p. 59 (Jan., 1910).

[6] Spencer, A. C.: *Economic Geology*, vol. viii, No. 7, p. 648 (Oct., 1913).

The Occurrence of Bournonite, Jamesonite, and Calamine at Park City, Utah

BY FRANK ROBERTSON VAN HORN, CLEVELAND, OHIO

(Salt Lake Meeting, August, 1914)

INTRODUCTION

IN June, 1911, the writer spent a few days in studying the economic geology of the vicinity of Park City. During this rather hurried visit a number of specimens of ore were collected, which have since been examined. The results of these investigations show that bournonite ($PbCuSbS_3$) is present in the ores in greater or less quantity. This fact has never before been reported, as far as can be learned. The failure to recognize this mineral is due to its similarity to tetrahedrite, with which it has probably been confused during a long period of years.

Jamesonite ($Pb_2Sb_2S_5$) has been reported from the region in a rather doubtful manner, but the writer has seen it in notable amounts from several mines, and can offer the first analysis of this mineral from the district.

Calamine ($H_2Zn_2SiO_5$) was found at one mine and has never been ecognized and reported from the district.

GEOGRAPHY AND GEOLOGY

Park City, Utah, is located at an elevation of about 7,000 ft. above sea level on the eastern slope of the Wasatch mountains. It is situated approximately 25 miles southeast of Salt Lake City. The geology and ore deposits have been described in detail by Boutwell and Woolsey.[1]

The mines visited are situated south-southeast from the town. The Silver King Coalition mine is located in Woodside gulch about 1 mile southeast of Park City, at an altitude of about 8,100 ft. The other mines visited were the Daly West and Quincy, which are in Empire canyon about 2 miles southeast of the town, at elevations of 8,300 to 8,500 ft. respectively. The Daly Judge tunnel goes into the mountain from Empire canyon about 1 mile south of the city, at an altitude of 7,700 ft.

[1] Geology and Ore Deposits of the Park City District, Utah, *Professional Paper No. 77, U. S. Geological Survey* (1912).

About two-thirds of the area of the district is covered by sedimentary rocks, and the remainder are of igneous origin. According to Boutwell,[2] the sedimentaries are divided as follows:

	Feet	
Ankareh	1,150+	Triassic
Thaynes formation (limestone, sandstone, shale)	1,190	
Woodside shale	1,180	
Park City formation (limestone, quartzite, sandstone, shale)	590	Permian (?)
Weber quartzite	1,350+	Pennsylvanian

The sediments were intruded from Cretaceous to Eocene by diorite, diorite porphyry, and finally andesite. It was these igneous rocks, especially the earlier ones, which are thought to have caused the formation of the ores of the district. Both previously and subsequently to the deposition of the ore, more or less faulting has taken place.

Occurrence of Ores and the Common Ore Minerals

The ores of the region are found both in fissure veins, and as replacement deposits in limestones. The two types are often associated, as in the Silver King Coalition and Daly West mines. The fissures have a general northeast-southwest trend. Most of the replacement or bedded deposits are found in the limestones of the Park City formation, although a few occur in the limestones of the Thaynes formation.

The commonest sulphide minerals, according to previous writers, are galena, tetrahedrite, sphalerite, and pyrite. To this rather short-list bournonite can be added. At one time the common occurrence of tetrahedrite in the region was doubted by the writer but now it is known that both tetrahedrite and bournonite are fairly common ore minerals. It is certain that in the past more or less bournonite has been called tetrahedrite, but which of the two minerals predominates is still a matter of conjecture. The chief oxidation products are cerussite and anglesite, with small amounts of malachite and azurite. According to Boutwell,[3] considerable amounts of bindheimite ($Pb_3Sb_2O_8+aq$) are found. This would seem to indicate that bournonite might be a more prominent constituent of the ores than tetrahedrite, because the former is a lead-antimony sulphide, whereas the latter is a copper-antimony compound, and would not oxidize directly to bindheimite without interaction of other substances.

One interesting feature which the writer has never before seen under natural conditions was observed on "coarse cleavable" galena from the Silver King Coalition mine. It was a polysynthetic twinning in two

[2] *Op. cit.*, p. 44. [3] *Op. cit.*, p. 114.

directions perpendicular to each other, although one set of striations was much better developed than the other (see Fig. 1). The twinning lines make angles of 45° with the cleavage cracks after the cube. In other words, the twinning plane is parallel to the diagonals through a cube face or to the secondary planes of symmetry, and consequently must be the face of a rhombic dodecahedron. For galena, this face is a gliding plane, and the phenomenon may be produced artificially by placing cubes of galena in a vise and applying pressure. A similar behavior of calcite is perhaps more widely known, where a proper application of pressure will produce artificial twin crystals parallel to the face of the negative rhombohedron $\frac{1}{2}R$. To the writer the presence of such striations on galena, indicates pressure after the deposition of the mineral, and the lines were undoubtedly formed by the pressures caused by the postmineral faulting

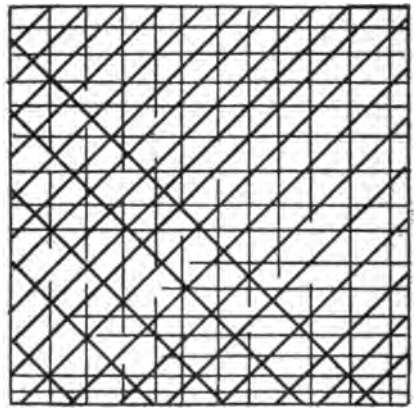


FIG. 1.—GALENA WITH CUBICAL CLEAVAGE PARALLEL TO THE EXTERIOR, SHOWING POLYSYNTHETIC TWINNING DUE TO PRESSURE ALONG THE CUBE DIAGONALS.

of the district. Faults of this character have been recognized in such mines as the Silver King and Daly West.

OCCURRENCE OF THE BOURNONITE

Bournonite is a very rare mineral, and has been reported previously from but three or four localities in the United States. The writer first saw a crystal of the mineral from Park City in the magnificent collection of the late Albert F. Holden, of Cleveland. A. T. Dalley, of the Silver King Coalition Mines Co., gave the writer a crystal which is now in the mineral collection of Case School of Applied Science at Cleveland. At the time, Mr. Dalley had in his possession another bournonite crystal, which he later donated to the Geological Department of the University of Utah, at Salt Lake City. It was also learned that the crystal in the

Holden collection was originally obtained from Mr. Dalley. The three crystals came from the 1,300-ft. level of the Silver King Coalition Mine and as far as the writer knows there are no others like them from the Park City district in existence. Mr. Dalley did not see the crystals in the mine and therefore does not know the exact conditions under which they were found. He has since seen other "smaller crystals that are in quartzite associated with other sulphides such as pyrite, galenite, sphalerite, and jamesonite."[4] Because of the rarity and size, the dimensions and weights of these crystals are given, as follows:

	Grams
No. 1. Holden Collection, now at Harvard University, $3\frac{1}{2} \times 3\frac{1}{2} \times 6$ cm.....	185.0
No. 2. Case School of Applied Science, $2 \times 2\frac{1}{2} \times 4$ cm.....	61.7
No. 3. University of Utah, $1\frac{1}{2} \times 1\frac{1}{2} \times 3\frac{1}{2}$ cm.....	44.4

Other specimens from the Silver King mine were given to the writer by the Superintendent, George D. Blood. In these the mineral was massive and was supposed to be tetrahedrite. However, most of the specimens contain lead in varying amounts up to 15 per cent., and are probably bournonite, as they resemble that mineral in color and luster. It is nevertheless possible that the massive mineral might be a plumbiferous tetrahedrite, which is a very rare variety, but has been found at several other places. Bournonite and tetrahedrite, when massive, resemble each other very closely. As far as the writer can see, the bournonite appears to possess a more brilliant metallic luster. The color is blackish lead-gray and the streak is black. In tetrahedrite from Park City, the luster is duller; the color has none of the lead-gray appearance, but is more like iron-black; and the streak is distinctly reddish brown instead of black. Possibly more than one variety of tetrahedrite occurs in the district. Chemically it is also difficult to distinguish the two minerals, especially if the tetrahedrite is plumbiferous, which the writer believes to be the case at Park City, because both minerals might contain lead, copper, antimony, and sulphur. In general, however, lead should predominate in the bournonite, and copper in the tetrahedrite. A quantitative analysis is necessary to determine the two minerals from a chemical point of view.

Mr. Talbot, the Superintendent of the Daly West mine, gave the writer several specimens of so-called "tetrahedrite" from the lower levels (1,000–1,200 ft.). One of these showed crystals on which we could not recognize any of the forms of tetrahedrite, but which seemed to possess an orthorhombic habit. These crystals were quite rough and much distorted, and seemed to disappear into a massive mineral, which had the same color and luster as the crystals, and which was in our estimation certainly the same mineral as the crystals. Toward the center of the

[4] A. T. Dalley: Personal communication, Oct. 9, 1913.

specimen pyrite in pentagonal dodecahedrons was found, and this was followed by coarse cleavable galena. Portions of some crystals and the adjacent massive mineral were broken off as carefully as possible, and analyzed by Dr. W. R. Veazey, of the Chemical Department of Case School of Applied Science, with the following results:

	Theoretical Composition of Bournonite Based on Formula $PbCuSbS_3$	Analysis of Daly West Mineral	Combining Weights	Ratio
Pb	42.54	43.18	0.209	1.009
Cu	13.04	13.14	0.207	1.000
Sb	24.64	25.03	0.208	1.004
S	19.77	19.59	0.611	2.951
	100.00	100.94		
		(Sp. gr., 5.829)		

An inspection of the results obtained shows conclusively that the mineral is bournonite, and that the ratio obtained from our analysis is very close to the theoretical composition required by the formula $PbCuSbS_3$. The specific gravity, 5.829, also conforms well to the average density obtained from specimens of the mineral from other localities. In all specimens seen by the writer the mineral was associated with pyrite and coarse cleavable galena, and was evidently formed later than either.

Occurrence of the Jamesonite

Jamesonite is another rare mineral, and has been found at but two or three localities in the United States. In the very able monograph on the Park City district, Boutwell[5] says: "Three specimens of a crystalline gray metallic mineral were found on the California dump. . . . . The material was too meager to permit thorough chemical determinations, but Dr. Hillebrand obtained qualitative tests for sulphur, antimony and a salt of lead, and accordingly is inclined to regard the mineral as jamesonite or warrenite." Under "Bournonite" the present writer has mentioned a personal communication from A. T. Dalley which shows at the Silver King Coalition mine an association of jamesonite with bournonite, galena, pyrite, and sphalerite. In the same letter Mr. Dalley says: "The best jamesonite (fibrous) occurs at the Daly Judge mine with (most commonly) pyrite, occasionally pyrite and galenite." In the Case collection are notable amounts of a mineral from the Silver King Coalition mine. It occurs in capillary crystals which are woven together into a felt-like mass. The mineral has a dark lead-gray color, a brilliant metallic luster, and a black streak. In one specimen the mineral rests upon a

[5] *Op. cit.*, p. 109.

perfect striated cube of pyrite. In three other specimens the jamesonite rests upon coarse cleavable galena, which in turn incloses pyrite. It would therefore seem that the jamesonite was the latest mineral to form. The felt-like mass was found to contain many small crystals of pyrite, which were carefully isolated so that an analysis of the capillary crystals could be made. The results obtained by Dr. W. R. Veazey were as follows:

	Theoretical Composition of Jamesonite Based on the Formula $Pb_2Sb_2S_5$	Silver King Mineral
Pb.....	50.84	55.75
Sb.....	29.46	21.34
As.....	00.00	4.14
S.....	19.70	18.43
	100.00	99.66

Our analysis shows that the mineral investigated conforms in general to the theoretical composition of jamesonite. It was found on comparing 38 different analyses[6] that there is a wide variation in the constituents between certain limits. The lead in our analysis is higher than the average, although one analysis of material from Russia contained 63.61 per cent. One feature of the Park City mineral, which is not common in most jamesonite, is that part of the antimony has been replaced by arsenic. It would therefore seem that a small amount of dufrenoysite ($Pb_2As_2S_5$), which is generally regarded as isomorphous with jamesonite, must be present in the Silver King mineral. In order to compare this analysis with most jamesonite analyses, the arsenic was converted into antimony (4.14 As = 6.63 Sb), after which the combining weights and molecular ratios were obtained as follows:

		Combining Weights	Ratio
Pb.....	55.75	0.270	$1.16 \times 2 = 2.32$
(Sb, As).....	27.97	0.233	$1.00 \times 2 = 2.00$
S.....	18.43	0.575	$2.47 \times 2 = 4.94$

It will be seen that the ratio of the recalculated analysis conforms quite closely to the theoretical formula $Pb_2(Sb,As)_2S_5$. If the original analysis be compared with the theoretical composition of dufrenoysite on one side, and that of jamesonite on the other, it will be found that, in general, the Park City jamesonite stands between the two, and is therefore probably an isomorphous growth of the two minerals.

[6] Hintze: *Handbuch der Mineralogie*, pp. 1031, 1032.

	Dufrenoysite, $Pb_2As_2S_5$	Park City Jamesonite, $Pb_2(Sb, As)_2S_5$	Jamesonite, $Pb_2Sb_2S_5$
Pb	57.1	55.75	50.84
As	20.7	4.14	0.00
Sb	00.0	21.34	29.46
S	22.2	18.43	19.70
	100.0	99.66	100.00

The only discrepancy is in the sulphur, which is a trifle low. It is also to be remembered that our analysis reached a total of 99.66 per cent. and was not calculated on the basis of 100 per cent.

Occurrence of the Calamine

Calamine ($H_2Zn_2SiO_5$) has never been reported before from the Park City region, as far as can be ascertained. The mineral was found on the dump of the Quincy mine as a drusy coating on cavities and pores. Several specimens were collected. The calamine was always associated with a brownish manganese oxide resembling wad. It was evidently entirely manganous oxide, as the usual manganese reactions were not observed until the mineral was oxidized with niter. The calamine crystals were grouped in the characteristic sheaf-like masses, which were in some places assembled into botryoidal and mamillary imitative shapes. The crystals were all thin tabular after the brachypinacoid $b(010, \infty\breve{P}\infty)$, which was striated vertically. The prism $m(110, \infty P)$ was present as small planes, as were also a macro- and brachy-dome [evidently $t(301, 3\breve{P}\infty)$, and $i(031, 3\breve{P}\infty)$]. The basal pinacoid $c(001, 0P)$ was not observed, and if present was very small. Although the writer was perfectly convinced as to the identity of the mineral, a qualitative chemical test was made for verification. The mineral was dissolved in hydrochloric acid and evaporated to dryness. It was taken up in hydrochloric acid and water, and boiled, after which the silica was filtered off. Nitric acid was added to oxidize any iron, and ammonium chloride was added to keep up any zinc. Ammonia was added to precipitate iron and manganese, and the precipitate was filtered. Hydrogen sulphide was passed through the filtrate, and a white precipitate of zinc sulphide was obtained, which fully corroborated the mineralogical determination.

The writer is unable to say whether the calamine exists in large amounts, but the discovery and associations here remind one very decidedly of the occurrence at Leadville, Colo., where large amounts of calamine and smithsonite have been recently found after having escaped recognition for many years. It is, of course, an oxidation product of sphalerite, and was probably associated with the usual minerals found in the upper

levels of the Quincy mine, such as cerussite, anglesite, malachite, and azurite.

Paragenesis and Conclusion

The common associated minerals, as well as the probable order of deposition, have been mentioned in the detailed discussion of the occurrence of the various minerals. The writer is, moreover, impressed with the fact that there is evidently constantly present in the ores of the Park City district, a lead-copper-sulphur-antimony series of minerals represented by galena, bournonite, tetrahedrite, and jamesonite. One gets the general impression that Park City is a camp mainly of argentiferous galena. The fact that an antimonial lead series of minerals is present is shown by the constant oxidation to bindheimite ($Pb_3Sb_2O_8+aq$), which is repeatedly mentioned by Boutwell. Study of the ores leads to the further conclusion that the sulph-antimonides bournonite, tetrahedrite, and jamesonite were all deposited later than the galena, and therefore that the lead-antimony solutions entered at a subsequent period. There are certain questions relative to depth which the writer cannot answer. The copper content of the Park City veins is evidently increasing with depth. Is this also true of the zinc? What relation have tetrahedrite and bournonite to each other? I have never found both minerals on the same specimen. What is the critical level of bournonite compared with tetrahedrite? Was bournonite more common on the upper sulphide levels, and has it simply escaped recognition during a long period of years? Is it being replaced by tetrahedrite in the lower levels as the copper content of the veins increases with depth? Or on the other hand is the bournonite now being found for the first time in the lower levels? These questions cannot be answered with the material and data at command. The suggestions are given in the hope that some of the questions may be answered by members who are more familiar with the district.

An Amendment to Sales's Theory of Ore Deposition

BY FREDERICK W. BACORN, BUTTE, MONT.

(Salt Lake Meeting, August, 1914)

THE paper of Reno H. Sales on Ore Deposits at Butte, Mont.,[1] is a careful and painstaking work, an important contribution to the literature of the subject. As is almost inevitable in a work of such magnitude, a few misstatements occur; and some of these I shall endeavor to point out. Mr. Sales has also fallen into error, as I think, in the application of his theory of ore deposition; and while the modification which I venture to suggest is a minor one geologically, it is of great economic importance. I refer to Mr. Sales's idea of an outward migration of mineralizing solutions from a supposed central copper zone. In order that I may make myself plain, I must first epitomize the theory as a whole, which I understand to be this:

All of the Butte ore deposits (save those along the Continental fault, not herein considered) are derived from solutions of magmatic origin, which, at the beginning of their ascent from a great depth, were acid in reaction, hot, and were carriers of copper, zinc, manganese and other metals. As they departed from their point of origin they lost heat, and, because of reactions with the wall rock, became more and more alkaline. The metal deposited from these solutions in their hot condition was principally copper; and, as they cooled, they deposited less and less copper and more and more manganese and zinc, until the copper practically disappeared.

With this theory in general I have no fault to find, nor have I, it must be stated, the qualifications necessary to enable me to pass upon it critically; but it seems to me to be reasonable and probable, and is one which I am quite willing to accept.

But Mr. Sales, for what seem to me to be insufficient reasons, has assumed that there is one deep central source of copper, relatively rich and small, from which all the solutions emanated. From this central source the solutions rose, through one channel or through a group of channels, to the surface, at and within a small area called the central copper zone,

[1] *Trans.*, xlvi, 3 to 109 (1913).

shown in Fig. 7 of Mr. Sales's paper, p. 58. From this central copper zone, via channels relatively near the surface, the solutions migrated concentrically.

In the veins within the first ring, called the intermediate zone, was deposited copper more or less intermixed with manganese, zinc, etc. Further out, in the peripheral zone, was deposited little copper with much manganese and zinc. Still further, beyond the peripheral zone, the depleted solutions made no noteworthy deposits.

I have spoken of the channels through which the solutions migrated outwardly as being relatively near the surface. Mr. Sales is not clear on this point; but I think his idea is that of a main trunk channel rising from the central source of copper, and tapped near the surface for horizontal distribution. If his idea were that of a radial distribution from the central source of copper, he would have said so; and, besides, a radial distribution, or any distribution from a point relatively near the central source, would not have produced the concentric arrangement which Mr. Sales conceives, within the small area considered in his paper. A circle 10,000 ft. in diameter takes in practically everything shown in the map on p. 58, yet if the central source were 20,000 ft. deep, no point in the circle would be more than 615 ft. further away from the central source than any other point, and 615 ft. would be relatively insignificant. There is no radial arrangement of the veins themselves, either on strike or on dip.

I think that I have stated Mr. Sales's views correctly, though perhaps not with technical accuracy. His views on this point, as I so understand them, I believe to be erroneous.

In the first place, I do not think that there is any "concentric zonal arrangement" of the district. There may be a sort of concentric arrangement about the area which Mr. Sales calls the "central copper zone;" but this concentric arrangement, in a view of the district as a whole, is a local phenomenon, and it may be one of a plurality of such.

Within this central copper zone are the great copper mines of the district. The veins, as explained by Mr. Sales, carry little or no manganese or zinc. Ordinarily they do not outcrop, and their apices, when exposed by surface cuts, show only iron-stained, crushed, altered granite, with stringers of iron-stained quartz.

West and northwest of this central zone is the "silver district," so called because here were the first mines of Butte, wherein mining operations were for silver only. North of the central copper zone is an eastward projection, in the shape of a wedge, of the silver district. The veins within the silver district ordinarily outcrop strongly, the outcrops being principally quartz, stained black by manganese. In depth the veins carry manganese and zinc; and until recently they were supposed to carry little or no copper.

From the central copper zone into this silver district, there is a pro-

gressive change from veins of the characteristic "copper" type to veins of the characteristic "silver" type; but eastward and southward from the central zone, this progression, if it exists at all, is by no means so obvious; and discoveries in these directions are coming so rapidly that it is safest to say that conditions there are unknown.

Furthermore, on going northward, across the wedge-shaped projection of the silver district, the order of progression is reversed; and one comes again into a region of "copper" veins; that is to say, of veins which ordinarily do not outcrop, which do not show the prominent black outcrops of typical "silver" veins, and which show little or no manganese or zinc; but which, on the other hand, show the typical crushed and altered granite and iron-stained quartz of the "copper" veins.

Here is a large section within which veins of the "copper" type predominate; and for specific illustration there may be pointed out at Mountain View Junction, only a little over a mile from the great Badger copper mine, many large, strong veins, which completely fulfill Mr. Sales's description of "copper" veins, and which, so far as developed, show no evidence whatsoever of inferiority.

The point of all this is, that while Mr. Sales is careful to state that the limits of pay ore are as yet unknown, his theory of the outward migration of the mineralizing solutions from the central copper zone implies to the outlying and undeveloped territory some degree of inferiority. His attention being not unnaturally focused on the ground which comes under his daily observation, he has unconsciously assumed that the present absence of operating copper mines in the outlying sections is proof of the absence of copper; and on this assumption he has predicated the idea of the concentric arrangement of the district.

As a matter of fact, Butte is peculiar among all the mining districts of the world in that development and growth have extended outward from the point where the first discovery of profitable copper ore was made; not, as elsewhere when bonanzas are discovered, by leaps and bounds, but slowly and haltingly, usually from vein to vein and from claim to claim. The reasons for this have been sociological, and have had nothing to do with geology.

But to continue the discussion, not only does it appear that any concentric arrangement, if it exist, is a local phenomenon, but there seems to be no other good reason to support the theory of outward migration.

The deep central source of copper is, of course, purely hypothetical. Nor is it necessary thus to account for the copper in the Butte mines, as Mr. Sales suggests (p. 80). Hundreds of analyses of Butte granite, made in the course of mining litigation, have disclosed the presence of copper. One specimen taken from a point without the central copper zone and particularly studied by Mr. Weed[2] carried copper to the extent

[2] *Professional Paper No. 74, U. S. Geological Survey*, pp. 93, 94 (1912).

of 0.006 per cent. A rough calculation shows that, at this rate, every 3 or 4 cubic miles of rock contains as much copper as has ever been mined in Butte; and it cannot be doubted that very many cubic miles of rock were drawn upon in the process of the mineralization of the district.

Much importance is attached by Mr. Sales to the porphyry dikes, although he does not clearly specify the part which he believes them to have taken. I think it will be found that these dikes are much more common phenomena than now believed to be, for on the surface they are ordinarily not conspicuous, and nobody has hunted for them; but even assuming that none exist save those described by Mr. Sales, their connection with the ore bodies is not obvious. They appear without as well as within the central zone; copper is found near them and thousands of feet distant; they are not permeable and were not channels for the solutions; they do not carry copper; and the veins are not enriched where they cut through the dikes. The only evidence of their relation to the ore deposits is in the fact that one of the series of ore-bearing fissures roughly parallels them. Standing alone, as it does, this fact would seem to mean only that the rock stresses which formed the porphyry-filled fissures persisted until after the Anaconda vein fissures were formed. Later, different rock stresses caused another series of fissures, not parallel with the dikes, but forming with them an angle of 45°, and these fissures, like the Anaconda fissures, are ore bearing, so that the evidentiary value of the parallelism between the dikes and the Anaconda fissures is greatly weakened.

From this review, it would seem that the evidence of the outward migration of the mineralizing solutions is very slight; and, on the other hand, there are certain facts which are incompatible with that theory.

The Jessie and the Edith May veins run outwardly from the central zone into the intermediate and peripheral zones; and hence, if there has been an outward migration, these veins must have been among those channels through which the flow took place. In such cases, we should expect to find, as we depart from the central zone, that the ore deposits would partake more and more of the characteristics of the silver veins, or, for short, that they would be more and more zincky. At any given point, the character of the deposit must be more zincky than at any point nearer the central zone, because the solutions, once cooled, would not regain heat, nor, once released from pressure, enter channels under greater pressure. What we do find is stated by Mr. Sales on p. 1581; that is, a zonal arrangement on a small scale. "High-grade chalcocite-enargite-bornite ores form the central part of the ore shoot with but little sphalerite and chalcopyrite and shade almost imperceptibly into zincky ore with chalcopyrite toward the ends of the shoots."

Now this is exactly what we should expect to find if the mineralizing solutions had come up through the open portions of the veins, there depos-

iting the copper minerals, and in the thinner places cooling off and depositing zincky minerals.

Again, the Badger vein, at the surface and for the first 1,000 ft. in depth, is what would be called a silver vein. At about the 1,000-ft. level copper begins to come in; and from the 1,300-ft. level to the present bottom of the mine is one of the finest bodies of copper ore in the district.

No part of this vein is within about 4,000 ft. of the central zone; and it is a practical impossibility to figure out any connection between it and the central zone which would account for the facts on the theory of the outward migration. This vein, unlike the Jessie and the Edith May, does not run from the central zone outward, but strikes eastward and westward. It does not approach the central zone on its dip; but is approximately vertical, whereas the principal veins in the central zone dip to the south; that is, away from this vein.

Again, as in the case of the Jessie and Edith May veins, the facts observed in the Badger vein are compatible only with the hypothesis that it was mineralized by solutions rising through the vein itself.

Finally are the veins at Mountain View Junction above referred to. In these the granite filling has been altered, as in the copper veins elsewhere, and at one point there is a considerable showing of copper carbonates, within the walls of and unmistakably a part of one of the veins. In addition to this, nearby in the railroad cut is an exposure, for 500 ft., of altered granite similar to or identical with the altered granite of the central copper zone.

What produced this alteration and what deposited this copper? Meteoric waters do not make such alterations (Sales, p. 1553). This place is 2 miles distant (northeasterly) from the central copper zone; and between are great areas of unaltered granite, zinc mines, silver veins, and all the phenomena of the "peripheral zone." There is no north-and-south fissure which could have carried the mineralizing solutions from the central zone. There is no flat, blanket-like fissure, known or suspected, which might have acted as a channel; and if, for the sake of this discussion, we assume the existence of such a blanket fissure, then we are confronted with the difficulty that where tapped by the Black Rock vein, a mile from the central zone, the solutions rising through the Black Rock vein made deposits characteristic of "silver" veins; and where tapped in the neighborhood of Mountain View Junction, 2 miles from the central copper zone, the solutions have effected results characteristic of the "copper" veins.

We are, therefore, driven to the conclusion in this case, as in the cases of the Badger, the Jessie, and the Edith May veins, that the solutions came up through the veins themselves, from whatever may have been their source.

There are a great many veins in the outlying territory which show no

variation (material to this discussion) from the veins mentioned: and as the process of reasoning which has been applied to a few veins is applicable to each of the others, it must be that all veins which were strong enough and open enough, acted as channels through which the mineralizing solutions found their way from great depths to the surface.

There is nothing unreasonable about this view. The Boulder batholith, of which the Butte district is a part, was a mass of lately molten granite (so Mr. Sales thinks) thousands of square miles in area and of great depth. There was a solid crust overlying a solidifying mass. Within the comparatively small area of the Butte district the crust was much broken and fissured. Steam and water were being forced to the surface. All openings must have acted as channels; and the fact that many fissures now show the effects of the hot water ("solutions") which flowed through them, is exactly what we might expect.

As will have been inferred from the foregoing discussion, the amendment to Mr. Sales's theory or hypothesis which I venture to propose, is this: that instead of there having been an outward migration of the mineralizing solutions, during which outward migration the cooling and the chemical changes took place, the flow of the solutions was generally upward, and the cooling and the chemical changes took place during the upward flow.

I see no difficulties in the way of accepting this modification. I do not see why the entire series of changes in the chemical character of the solutions, as well as the loss of heat, described by Mr. Sales, may not have taken place in each vein. I believe it possible to explain on this hypothesis all the phenomena described by Mr. Sales. This I can myself attempt only in a very general way; and it is to be hoped that Mr. Sales and other geologists will give the matter further attention.

I think the phenomenon of the central copper zone may be explained in the following manner:

Here are many veins, including a number belonging to the Anaconda system. The Anaconda fissures were the earliest, and the solutions rising through them, because of the long period during which they acted, or because of their greater concentration, or perhaps of their greater pressure and heat, or perhaps because of all these conditions, altered the granite to a greater extent than has yet been observed elsewhere in the district.

There is little zinc or manganese in the central zone. If the deposition of these metals depended on the lowering of the temperature of the solutions to a certain critical point, then we have only to assume that the temperature of the solutions in this region remained above that critical point until they reached the surface, or until they rose higher than the present surface. The larger the volume of solution, or the faster its flow, the further up would its temperature be maintained.

In the case of the Emily vein, in the Badger mine, we have another

condition, and we may say that most of the copper contained in the solutions was deposited by the time they had reached the 1,300-ft. level, in the form of chalcocite and enargite. A little copper was left, and this was deposited as a fringe, perhaps 300 ft. deep, above the enargite and chalcocite, in the form of bornite and chalcopyrite. A similar fringe, in a number of other cases, is reported by Mr. Sales. Above the copper, the temperature of the solutions was lowered to a point permitting the deposition of the minerals characteristic of a "silver" vein. Now if the critical point for the deposition of the "silver vein" minerals had been a few hundred feet deeper than it actually was, or if the Emily vein had never been explored to a depth greater than 1,000 ft., then the Emily would now be regarded as a typical "silver" vein. Quite likely all "silver" veins are copper veins in depth, though it may often be the case that the point at which copper comes in is too deep for commercial exploitation.

In some veins are large copper ore shoots which do not extend to the surface, above which copper there are no notable deposits of "silver vein" minerals. Perhaps these cases can be explained on the theory that the copper in the solutions was exhausted at the tops of the copper ore bodies; and that the temperature of the solutions, at these points, had not fallen to the critical point for the deposition of the "silver vein" minerals. From the top of the copper, up to the points where the temperature so fell, which points might be above the present surface, the veins would be practically barren, showing only an alteration of the granite.

These are matters which must be worked out in detail by qualified men; and it is greatly to be hoped that not only will Mr. Sales himself be able to give further attention to the work which he has so well begun, but that others will enter the field, where much remains to be done.

People coming to Butte from the outside may well be surprised to discover how much does remain. In view of the many years during which Butte has been known as a "bonanza camp," in view of the enormous production of copper, which continues without diminution, it is indeed strange that almost all knowledge of the district is confined to the small area, perhaps 2,000 acres, which Mr. Sales has described in detail.

Dip Chart

BY HOWLAND BANCROFT, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

THE writer has observed that some confusion is experienced by many mining engineers in making vertical sections of ore deposits in mining properties which they have examined. Having much examination work calling for the correct interpretation of vein structure and the subsequent

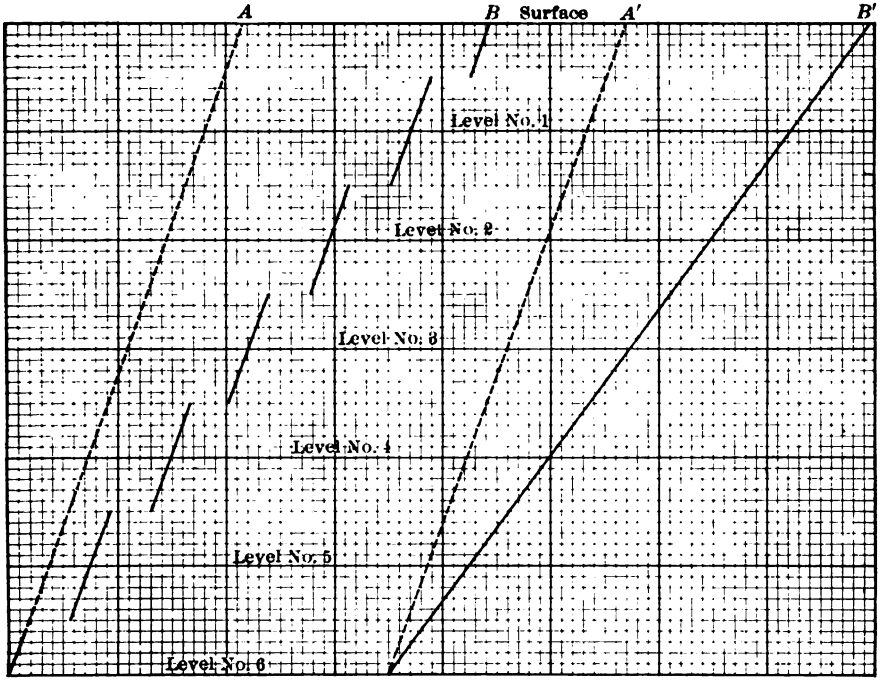


FIG. 1.—CORRECT AND INCORRECT WAY OF PLATTING A VERTICAL SECTION OF A VEIN.

platting of the same on sections, it has occurred to the writer that the accompanying diagram may be of use to others who at times may be called upon to prepare numerous sections. For example, a great mass of data on strike and dip of ore bodies is collected underground in separate

shafts, on different levels, drifts, cross-cuts, raises, winzes, etc. In assembling these data and preparing vertical sections the various dips of the deposit are frequently platted just as recorded underground. The result is that a deposit, which may be continuous from the surface to the lowest level, appears on the vertical section as a series of disconnected fissures having dips ranging from 0° to 90°. This is due to the fact that observations are taken in various parts of the mine and the plane of the vertical section is rarely, if ever, at right angles to the strike of the vein throughout the course of its dip plane.

Fig. 1 represents the correct and incorrect way of platting a vertical section of the same vein where observations have been taken at different levels on a vein having a uniform dip. In this ideal figure it is presumed that each level intersects the vein at the same angle. *A* and *A'* represent the platting of a vein having a uniform dip of 70° on a section the plane of which is at right angles to the strike of the vein. *B* represents the same vein incorrectly platted on a section the plane of which intersects the strike of the vein at an angle of 30°. Observations of the dip and strike of the vein were made on each level and the vein platted with its actual dip. *B'* represents the same vein correctly platted on a section the plane of which intersects the strike of the vein at an angle of 30°.

While the formula for determining the apparent dip of a vein to be platted on a vertical section may be known to many, this method of determining apparent dips involves a very tedious series of computations if the sections are numerous. The accompanying dip chart, Fig. 2, is submitted with the hope that it may save much laborious calculation in preparing vertical sections. This diagram is not original with the writer. D. W. Brunton devised a somewhat similar method for his own use at Butte, about ten years ago, and he now has in the process of manufacture "Brunton's Slope Chart," a 9 by 10-in. cardboard diagram with a swinging arm which can be read to a quarter of a degree and is complete for dips ranging from 0° to 90°. This will be entirely satisfactory for office use.

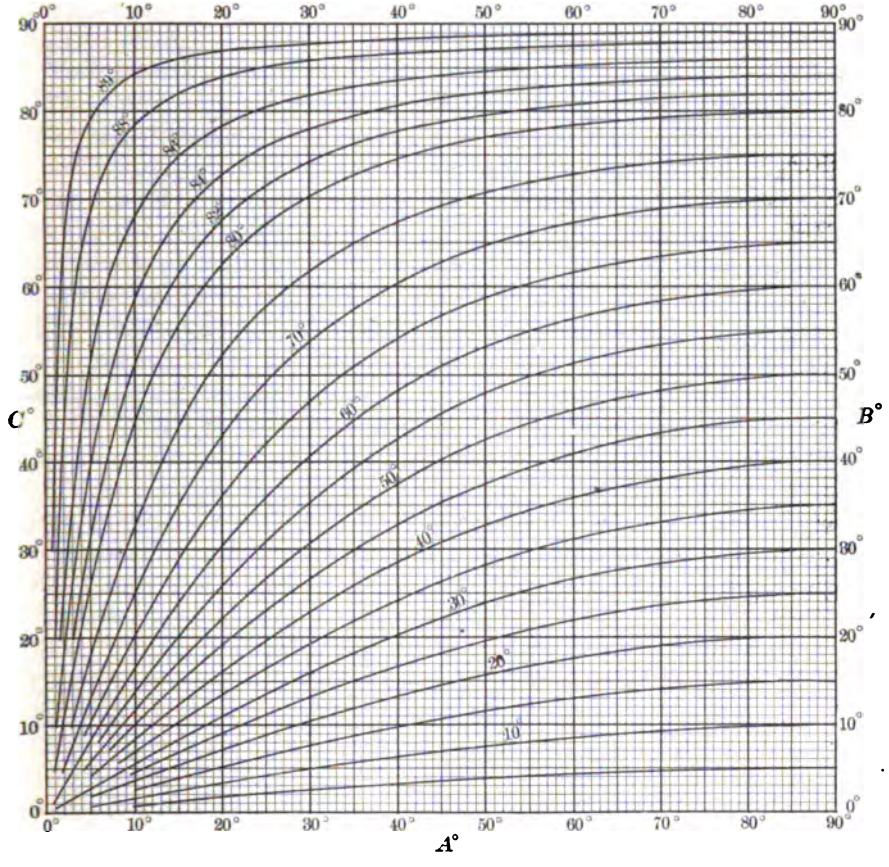
In 1912, D. F. Hewett published[1] a brief description of A Graphic Method for Dips on Geologic Sections. This was accompanied by a diagram, 7 by 7 in., on thin paper, which showed graphically the determinations for platting dip angles differing by 5°. Mr. Hewett's chart is available only to readers of *Economic Geology*. Furthermore, it is too large for field use, and the dip angles have 5° intervals.

The accompanying dip chart, prepared by A. B. Crosley, is of such size that it can conveniently be placed in the back of the ordinary field notebook, and the intervals between angles of dip of veins from 80° to 90° are only 2°. Hence, intersections with the vertical section for veins

[1] *Economic Geology*, vol. vii, No. 2, pp. 190, 191 (Mar., 1912).

dipping over 80° can be more accurately platted with this diagram than by Mr. Hewett's.

Mr. Brunton's slope chart is more accurate than either Mr. Hewett's or the diagram here presented. However, it is not practicable to carry

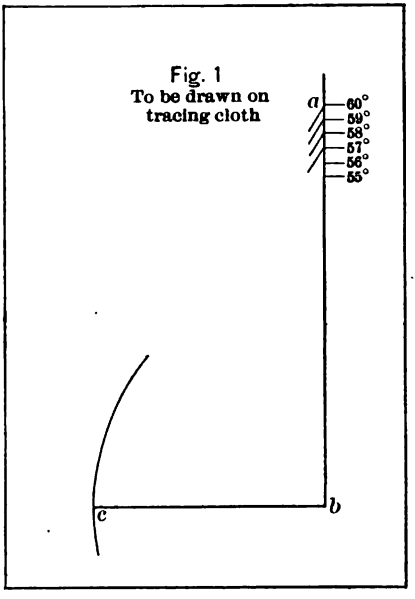


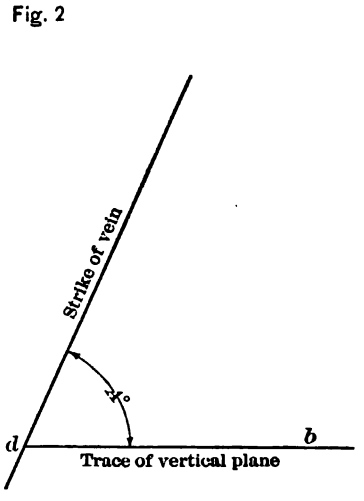
A Vein with Dip B°, Intersecting a Line of Section at an Acute Angle A°, is Platted on the Section with an Apparent Dip C°. Formula: Tan C° = Sin A° Tan B°.

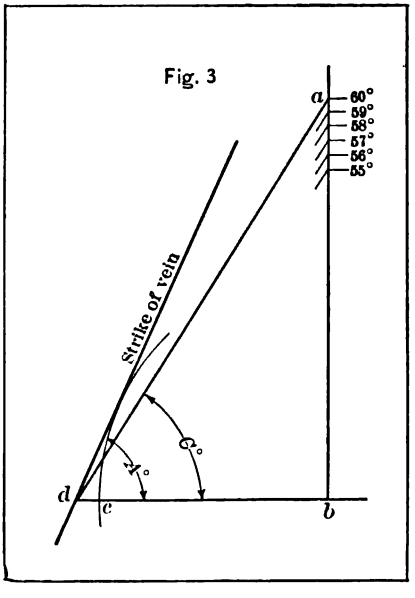
Fig. 2.—Dip Chart.

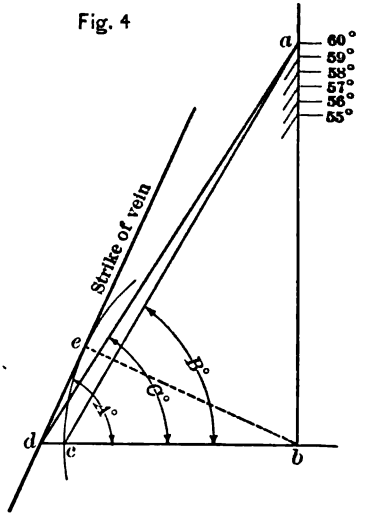
Mr. Brunton's chart in the field, its use being confined primarily to the office.

In using this diagram it is to be remembered that the angles indicated at the bottom (*A*) represent the difference in strike of the vein it is desired to plat and the plane of the vertical section. The curved lines represent dips of veins (*B*). The points of intersection of the vertical lines (above *A*) with the curved lines (true dip lines *B*) at the horizontal









FIGS. 1 TO 4.—METHOD OF USING DIP CHART.

lines (C) are the apparent dips at which the veins should be platted on the vertical section. For example, the strike of a vein dipping 86° intersects the plane of a vertical section at an angle of 25°. It should be platted on the vertical section with an apparent dip of 80° 30′.

Discussion

Theodore Simons, Butte, Mont. (communication to the Secretary*). —While preparing maps and models for use in mining litigations the writer was frequently confronted with the problem referred to in Mr. Bancroft's paper. His chart gives the apparent angle of dip with which a vein should be plotted on a vertical section not at right angles to the strike of the vein. For a quick determination of this apparent angle the writer has used an exceedingly simple graphical method, which he believes may prove helpful to engineers engaged in practical work. It is submitted, not as a criticism of Mr. Bancroft's paper, but as a means of solving the same problem when no chart is available and when calculations would consume too much time.

The writer's method requires merely a few lines marked off on a piece of tracing cloth, as shown in Fig. 1: Draw line bc on any convenient scale. At b erect a perpendicular, ba. By means of a protractor cut off on ba a number of angles within the probable range of variation of dip, so that a line connecting a and c, for instance, forms an angle of 60° with bc, etc. With b as a center and bc as a radius, draw a circle through c. This completes the tool for solving the problem.

Fig. 2 represents the line of strike of the vein and db the trace of the vertical plane on which the vein is to be plotted with the apparent angle of dip. $A°$ is the angle which this plane makes with the strike.

Fig. 3 shows the application of the method: Place line cb of tracing over line db of Fig. 2 and move fore and back until line of strike becomes tangent to the circle. Connect d with the point (a) that marks the true angle of dip as found by measurement in the mine (60° in the case illustrated). The angle adb is the apparent angle of dip ($C°$) to be used in the vertical section.

Proof.—As stated in Mr. Bancroft's paper, the apparent angle of dip is:

$$\tan C° = \sin A° \tan B° \qquad (1)$$

From Fig. 4, which is the same as Fig. 3 with a few lines added for sake of demonstration, we have:

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* Received July 28, 1914.

$$\frac{ab}{bd} = \tan C^\circ \text{ whence } ab = bd \tan C^\circ;$$

also

$$\frac{ab}{bc} = \tan B^\circ \text{ whence } ab = bc \tan B^\circ;$$

whence $$bd \tan C^\circ = bc \tan B^\circ;$$

or $$\tan C = \frac{bc}{bd} \tan B^\circ \qquad (2)$$

From Fig. 4, $$\frac{bc}{bd} = \frac{be}{bd} = \sin A^\circ$$

Introducing in equation (2) we get:

$$\tan C^\circ = \sin A^\circ \tan B^\circ \qquad (3)$$

This is the same as equation (1) and shows that angle C° in Fig. 4 is the required angle.

FRANK A. LINFORTH, Butte, Mont. (communication to the Secretary*).—Referring to Howland Bancroft's paper, entitled Dip Chart, and also to the various other charts and formulæ for the correction of dip in making geologic cross-sections, I would like to emphasize the importance of making these corrections and to call attention to the method used by the geological department of the Anaconda Copper Mining Co. for this work.

Cross-sections are used extensively in Butte for laying out development work in the mines, for checking the interpretation of geologic facts on the separate level maps, and to some extent in legal work. They are not generalized drawings, but accurate records based on observed facts, and great care is taken to make them geometrically as well as geologically correct. The complex structure of the Butte district with its east-west, northwest, and northeast systems of veins and faults renders necessary the corrections for dip on almost every cross-section. If a section is laid out at right angles to any one system of veins, the observed dips for that system can be platted without alteration, but, obviously, the observed dips for the other two systems must be corrected before platting if they are to appear properly on that section. There are many examples in the geologic work at Butte where it has been necessary to make accurate vein correlations between new or partly developed mine levels. In some of these cases there have been a number of veins to choose from and the correct correlations have come from the application of corrected dips on the sections. That these

* Received Sept. 21, 1914.

correlations are correct is invariably proved by later developments in the mines.

Formerly these corrections were roughly calculated. More recently D. F. Hewett's excellent chart referred to in Mr. Bancroft's paper was enlarged and used in the routine work of the office. As a wall chart, however, it was found to be less convenient than a tabulated arrangement of values in which the angles of intersection with the section appeared across the top of the sheet and the angles of observed dip were shown in the left-hand margin. The required value is found at the intersection of the vertical and horizontal columns under the proper headings. This is believed to be a better scheme than using the curves, but both are now supplanted by a small instrument devised and patented by the writer. This instrument, Fig. 5, is a symmetrically designed box 1¼ in. square, and about 8 in. long. A small square opening is

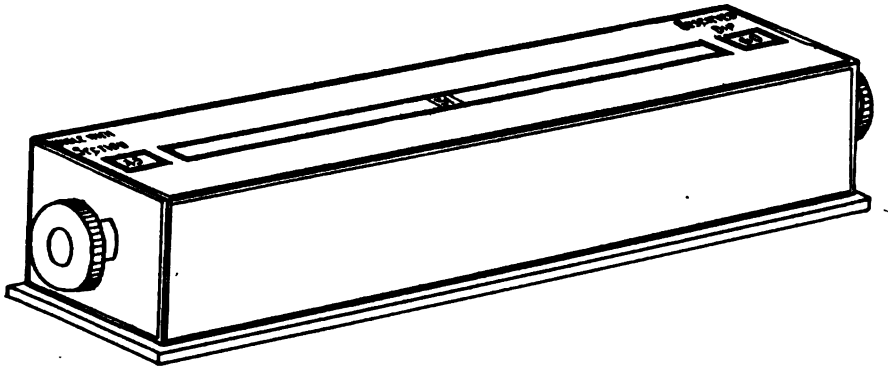


FIG. 5.—THE DIPOMETER.

cut in the top of the box near each end. One opening bears the designation, "Angle with section"; the other, "Observed dip." The observed values are made to appear in their respective openings by turning small knobs, one at each end of the box. Increments of 5° are used and this interval is found to be satisfactory for practical purposes, although in a slightly larger instrument this interval could be diminished. As soon as the observed values are set, the required value appears as a definite number of degrees in an opening cut in the top of the box. No other numbers appear and no chance for error is introduced, provided that the observed data are properly set up. All confusion is removed, and the determination is made much more rapidly than by tracing out a curve on a chart or figuring from a formula.

The construction of the instrument is very simple. The values of the required angle were figured from the various dips and angles with the section and were tabulated. The sheet of figures thus ob-

tained was mounted on a cylinder, and set in the box so that it could be easily rotated by one of the knobs. A cylindrical shell in which certain openings had been cut was placed over the first one, and its rotation about the common axis is controlled by the other knob. The indices were then adjusted so that the proper value appears for any given set of angles.

The small dimensions of the device, the ease of making the determinations, and the absence of chance for error make this method ideal.

JAMES UNDERHILL, Idaho Springs, Colo. (communication to the Secretary*).—In connection with Howland Bancroft's Dip Chart, no one seems to have called attention to the fact that the same formula and diagram can be used to determine the strike of veins from the direction of the outcrop on surface or to trace an outcrop when the strike and dip are known, as shown in Fig. 6.

It frequently happens that a vein whose dip and strike are known is encountered in a cross-cut tunnel and it is desired to know if some vein on the surface is the same. The direction of the outcrop in rough country may be many degrees from the strike of the vein. Or again, knowing the strike and dip it is often desirable to trace the extension of a vein or dike. For many years the writer has used the accompanying table, arranged to fit *Lefax Engineer's Loose Leaf Pocket Book.*

As an example, assume that the strike of a vein is due north and its dip 50° W. On the surface we find a vein, and the line of outcrop dips 15°. That is, the surface ground has a slope of 15° in the direction of the apex. From the table, the angle between strike and outcrop direction is found to be 13°, and as the dip is west the direction of the apex of the vein on the surface, or outcrop direction, should be N. 13° W. If the surface vein is running in this direction and is in about the right place we conclude that it is the same vein. The same results are obtained from Mr. Bancroft's chart by taking 15° on the horizontal line, left or right side, following line of 15° till it reaches 50° dip curve and then reading 13° on perpendicular line below.

Or again, an outcrop or vein apex on the surface has a course of N. 25° E. and dip 60° NW. The line of outcrop or surface of the ground where the vein apexes rises with a vertical angle of 30°. The vein, we find from the table, strikes 19° 28′ W. of the line of outcrop, or N. 10° 32′ E. From the dip chart as before we have 19° 30′ as the angle between the direction of the outcrop and the true strike of the vein.

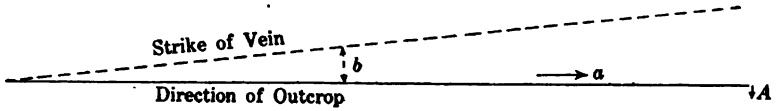
The same results may be attained with almost any solar attachment with which the writer is familiar, or with an ordinary top telescope of the Scott model.

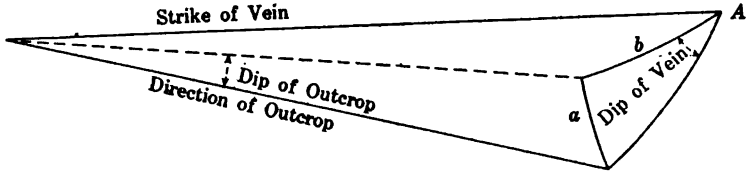
* Received Jan. 18, 1915.

VEINS—STRIKE DIRECTION OF OUTCROP

$$\text{Sin } b = \tan a \cot A$$

Sin angle between direction of outcrop and strike of vein is equal to tangent dip of the line of outcrop by cotangent dip of vein.





Dip Vein	Dip Outcrop							
	5°	10°	15°	20°	25°	30°	35°	40°
20°	13° 54′	28° 59′	47° 24′	90° 0′				
25°	10° 49′	22° 13′	35° 4′	51° 18′	90° 0′			
30°	8° 43′	17° 47′	27° 39′	39° 5′	53° 52′	90° 0′		
35°	7° 11′	14° 36′	22° 30′	31° 19′	41° 45′	55° 33′	90° 0′	
40°	5° 59′	12° 8′	18° 38′	25° 43′	33° 45′	43° 29′	56° 34′	90° 0′
45°	5° 1′	10° 9′	15° 32′	21° 22′	27° 48′	35° 16′	44° 27′	57° 3′
50°	4° 12′	8° 31′	13° 0′	17° 47′	23° 2′	28° 59′	35° 59′	44° 45′
55°	3° 31′	7° 5′	10° 49′	14° 46′	19° 3′	23° 51′	29° 22′	35° 59′
60°	2° 54′	5° 50′	8° 54′	12° 8′	15° 37′	19° 28′	23° 51′	28° 59′
65°	2° 20′	4° 43′	7° 10′	9° 46′	12° 34′	15° 37′	19° 3′	23° 2′
70°	1° 49′	3° 41′	5° 36′	7° 37′	9° 46′	12° 07′	14° 46′	17° 47′
75°	1° 21′	2° 42′	4° 7′	5° 36′	7° 10′	8° 54′	10° 49′	12° 59′
80°	0° 53′	1° 47′	2° 42′	3° 41′	4° 42′	5° 50′	7° 5′	8° 31′
85°	0° 26′	0° 53′	1° 21′	1° 50′	2° 20′	2° 54′	3° 31′	4° 12′

FIG. 6.—DIAGRAM AND TABULATION AS ARRANGED FOR LEFAX ENGINEER'S POCKET BOOK.

Mining Methods at the Copper Queen Mines

BY JOSEPH P. HODGSON, BISBEE, ARIZ.

(Salt Lake Meeting, August, 1914)

IN 1880, mining operations were commenced at the Copper Queen mine. The famous Queen orebody, which extended to the surface, was first quarried from a large open cut in the outcrop. The orebody was followed down to the 300 level with the Queen incline, and stoped. This, we believe, marks the introduction of square setting in the Bisbee district, which is still the method most generally used.

Orebodies

The orebodies in the Copper Queen mine occur in the limestones, and the main portion of the ore has been mined from the Abrigo, Martin, and Escabrosa limestones. While the orebodies outcrop in the extreme western end of the mine, the general dip is to the east and south, at an angle of about 20°, although this dip is not by any means regular, being, in fact, very irregular locally. The ore is remarkable for its variableness in character, some of it being very soft, requiring a large amount of timber, and other portions consisting of extremely hard sulphides. The orebodies, in general, while remarkable for their continuity, are very irregular, both as to shape and size. The mine having been in operation so many years, and mining having been prosecuted over such a large area, the operations have caused the shrinkage of the overburden, and there is a constant movement, which is shown by large cracks and interstices in the surface rocks, which largely contribute to the very high deadwork or maintenance cost of the mine. It is surprising that for year 1913 it cost almost $400,000 just for the one item of repairs and deadwork in this property.

Haulage and Hoisting

The underground openings have been extended in the neighborhood of 200 miles. The ore hoisting is performed at a centrally located shaft, the Sacramento, to which all the ores from different parts of the mine are conveyed by electric haulage, of which there was slightly over 9 miles in operation in 1913. The hoisting levels at the Sacramento shaft are placed 200 ft. apart, commencing at the 400 level and continuing down to the 1,600 level, the average hoisting distance being 1,000 ft. The

method of hoisting is by Kimberly skips, which are loaded from pockets, and as high as 400 skips have been hoisted through this shaft in a 7½-hr. shift.

Intermediate tramming to haulage chutes is done, in general, by mules and hand. Waste material is used for filling the square-set and cut-and-fill stopes, and when there is a surplus, it is sent to the surface at the subsidiary shafts, of which there are seven in operation, and which are also used for hoisting and lowering men, timber and supplies.

Lighting

All main haulageways, powder houses, etc., are lighted by electricity. The lights used by workmen have been candles, but carbide lamps are being substituted, as it is thought that by using them there is less liability of fire, and it has also been demonstrated that carbide is more economical. It is a better light to work by, and, as much sorting of ore is done, it enables closer sorting.

Compressed Air

All hoists, with the exception of the Sacramento, are operated by compressed air, generated at the central power plant, by compressors having a total capacity of 21,000 cu. ft. of free air per minute. Electric power for underground and surface lights and haulage is supplied by three Curtis turbo-generators, which are connected with seven 407-h.p. water-tube boilers.

Square Setting

Up to about 1½ years ago, square setting was used exclusively in this property. The system, as a whole, has been very successful. It is quite elastic, and permits the following of stringers from any point in the stope, and also permits of efficient prospecting. Perhaps the greatest objection to it in very soft ground, is the liability to loss of the stopes by excessive weight. There is also a high timber cost, and, as a whole, perhaps it is not as economical as other systems, although it must be said that a large proportion of the mine will always be worked upon this plan, by reason of the unequal and changing character of the ground.

The general custom in the square-setting system practiced here is to block the ore out in sections, numbering the sections consecutively, the object being to mine, if possible, four sections around one central raise. This can often be done, but frequently the ground is of such heavy character, and so much weight is thrown upon the timbers, that it is impossible to take out more than two or three sections to any one raise. These sections are laid out according to the local character of the ground, and are from two to four sets in width, and from six to ten in length.

Great care must necessarily be taken in laying out the work to avoid making sections too large and of too great width, so as to risk the possibility of caving.

As the stoping progresses from the sill upward, the raise is usually

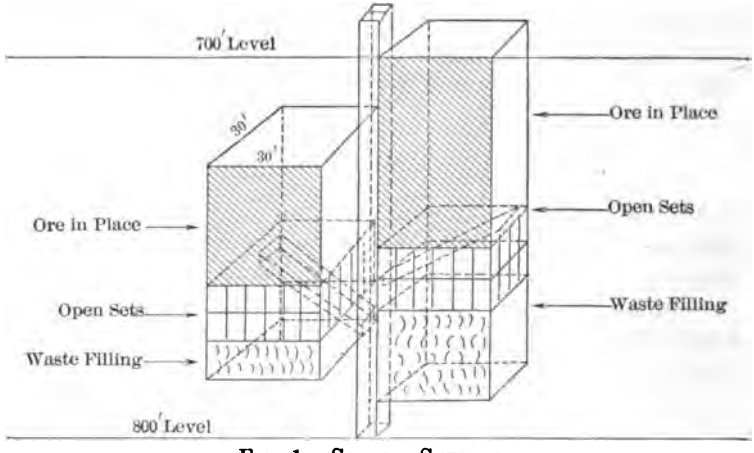


FIG. 1.—SQUARE SETTING.

sent through to the next level, to admit of the lowering of timber, the proper ventilation of the stope, and the dumping of filling; as it has been found necessary to carry the filling within about two floors of the back of

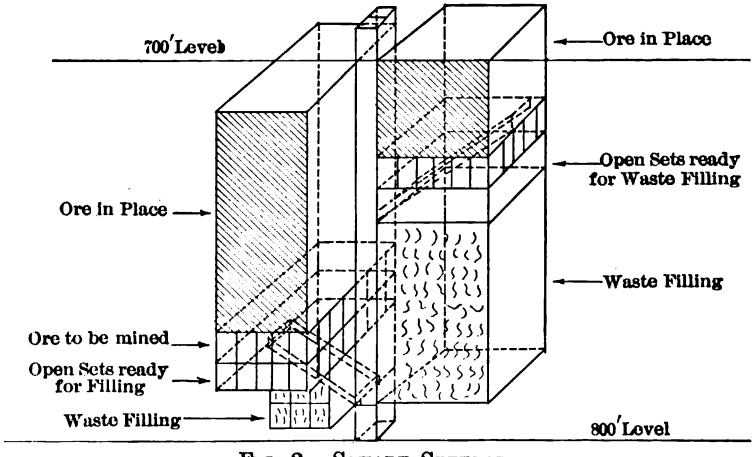


FIG. 2.—SQUARE SETTING.

the stope and immediately below where men are working. In many of the stopes, particularly in the oxide ores, a good deal of hand sorting is necessary, as it has been demonstrated that it is cheaper to eliminate the waste in the stope than to pay for smelting it. While this in many

cases increases the mining cost materially, yet the company believes it is good business to leave the waste in the gob rather than put it into chutes, tram, hoist, and pay transportation and smelting charges upon it. In order that nothing is mined but what shows a margin of profit, a system of minima is in effect, based upon the selling price of copper.

Figs. 1 and 2 illustrate different stages of extracting the ore by means of sections under the square-set system. The chutes are so arranged as to require the least amount of mucking.

Cut and Fill

Within the last two years, the management has been making some experiments in other mining methods, and, in certain portions of the mine, notably in the Holbrook, Spray, and Gardner divisions, some cut-and-fill stopes have been opened. This system of mining is, of course, applicable only in hard ground, and these stopes are exclusively in sulphide ores. The experiments have to date proved successful, having materially decreased the mining cost, as compared to the square-set method, and it is believed that this system should be worked wherever the conditions are suitable. The method in vogue is somewhat as follows:

The orebody is prospected as far as possible in advance, and the side and vertical dimensions of the ore are determined. Drifts are driven where possible under the bottom of the ore and raises put through the ore to the level above to permit the dumping of filling. Chambers are then cut out, drifts formed either by cribs or sets of timber, the back blasted down, raises cribbed up at convenient points, and filling dumped in, upon which the men may work and at all times be kept close to the back. Wherever possible, the stope is worked on an angle of about 45°, so that the broken ore may slide down upon a plank bed laid upon the filling to the chutes. This materially reduces the cost of getting the ore into chutes, and is very desirable wherever it is possible to use it. Prospecting can be done from any elevation, as the stope is worked up to that point, and the filling easily and cheaply disposed of. Wherever it is possible to work stopes upon an angle of 45°, very little timber, in general, is needed, as the slope of the ground materially helps to support the stope. Wherever a stope cannot be worked upon the slope and where the backs are carried horizontal, it is often necessary to put up temporary supports by cribbing and blocking up the back to make the stope safe while the ore is being extracted.

This system is also elastic, inasmuch as small blocks can be worked out wherever it is deemed necessary, provided the back is heavy and will not admit of being opened up in a fairly large chamber. One absolute necessity upon working this system is that the men be watched closely and taught to bar down the backs, taking care before they commence

drilling operations that all loose or unsafe ground is taken down. Perhaps a disadvantage is where ore is intersected by stringers or bunches of waste,

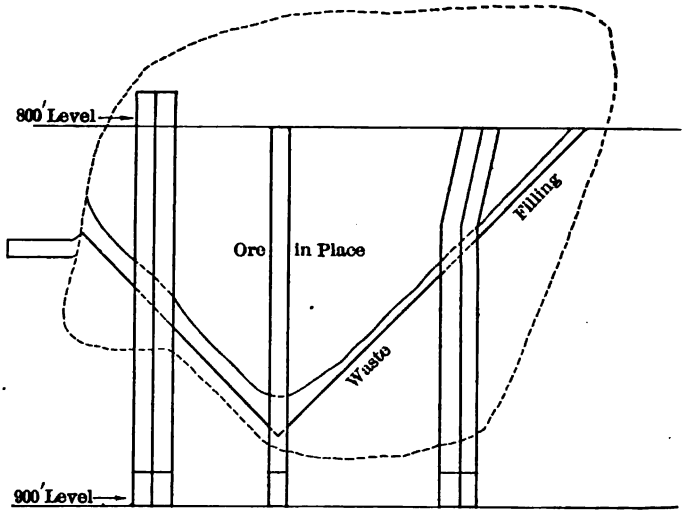


FIG. 3.—CUT AND FILL.

but, if care is taken, this waste can always be blasted down or put in the gob and the ore mined clean. The cut-and-fill method, up to the present time, has worked fairly successfully in the Copper Queen mines, and the

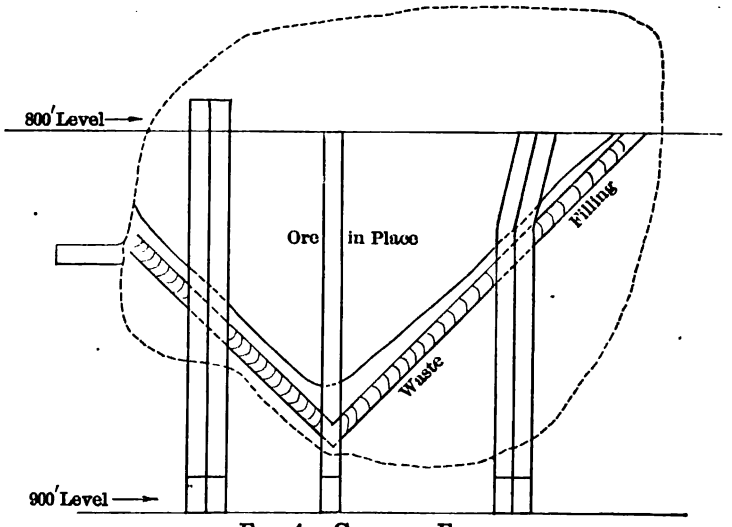


FIG. 4.—CUT AND FILL.

writer knows of many mines in Michigan and other places that have been worked successfully by similar means.

Referring to Fig. 3, starting at the top of the slope, water holes are drilled and a section of ground about 8 ft. thick and 20 ft. wide is blasted down. The work is done underhand wherever it is possible in order to keep the back solid.

Fig. 4 shows that the ore has been removed, waste filling has been run in and is floored over, consequently, the stope is again in the condition as shown in Fig. 3.

Shrinkage

Up to this time, only one place in the Copper Queen mine has been found where, in our judgment, a shrinkage stope could be developed. This stope is on the 1,100 level of the Lowell mine, and stoping is in progress there upon an orebody approximately 100 ft. in length and about 50 ft. in width.

In order to use the shrinkage system successfully, the surrounding character of the walls must first be ascertained, and it must be demonstrated beyond a doubt that they are strong enough to permit the removal of the ore after the stope has been carried to the top of the orebody or to the level above. This work is in progress at present in the Lowell mine and promises to show very material reduction in cost, as compared to square setting in the same character of ground. Practically no timber is necessary, and as the ore is kept close to the back the workmen are at all times close to the working face. As in the cut-and-fill system, care must always be taken that the workmen bar down and make safe the backs before commencing drilling operations. One disadvantage in the shrinkage system is that should bars of waste occur in the orebody, it must necessarily be mined and there is a danger of it becoming mixed with the ore.

Top Slicing

Another system that has also been receiving attention in these properties is that of the top slice. The top-slicing system probably originated in the iron-ore mines of the northern part of England, and, we believe, was first introduced in this country in the iron mines in northern Michigan. This system consists of first driving in the main-level drifts, crosscutting and finding the extent of the orebody, putting up raises through the orebody to the top of the ore, and commencing operations at the extreme top of the orebody. It must, of course, be demonstrated to the satisfaction of the management that there is no possibility of other orebodies being over the country that is to be mined, as the system, when properly used, does not necessitate filling.

The method simply consists of driving lateral drifts and taking out the ore in small blocks, making sure to clean the top of the orebody, placing either plank or split lagging upon the sill of every individual slice as the

operations are continued downward, thereby forming a mat, upon which the overburden and débris will rest. It is usually found in the preliminary operations of a top slice that the heaviest weight from the overburden is attained when the first three or four slices are being extracted. After this, the mat, old timber, and overburden become intermixed, and in a measure self-sustaining. This system is applicable and desirable in very soft or wet ground, and it is the writer's opinion that in this character of ground it will prove successful and profitable where square setting and other methods would fail.

Top slicing has been commenced in what is known as the Dividend slice of the Czar mine. This orebody contains perhaps 750,000 to 1,000,000 tons of very soft, wet, aluminous ore, and wherever square setting had been tried, it was found to be very expensive and almost impossible to complete a section successfully. As a preliminary to starting this slice, a drift has been driven in the foot wall on the 400, the orebody lying on the foot wall and extending about 50 ft. above the 200 level. Raises were then put up to the 200 level in the foot wall, as it has been found that raises in the orebody will not stand the terrific pressure which is brought to bear upon them.

On account of the ore being very wet and aluminous, it is very hard to handle in the chutes, and concrete pockets have been designed which have the shape of a funnel, with the large portion of the funnel downward. About 30 ft. above the 400 sill, or in the top of this funnel, an offset or baffle has been put in, and from this point the raise is continued to the 200 level. This raise is circular and lined with concrete, and, while we have not yet proved that this type of pocket will be successful, we are confident that it will materially lower the cost of handling the ore.

The work in this orebody has not progressed to a point where a comparison of costs can be made, but we are sure that the operation will be successful.

One advantage of top slicing is that it is very elastic. Drifts for prospecting may be driven in any direction from any floor, and the waste disposed of in the workings. Another advantage is that in mining the orebody from the top down, the ore is mined clean, and still another is that, wherever it is desirable, some incline raises may be put up at any point from main raises to the mining floor to lower the cost of tramming. Several orebodies in the Czar, Holbrook, Gardner, and Sacramento mines are being developed upon this plan, and the management is of the opinion that they will show a material lowering of costs, as compared to square setting. It must be understood, however, that top slicing can only be used where it will not damage any portion of the mine, and, particularly, it must be demonstrated, as before noted, that there are no orebodies above the territory worked upon this system.

Fig. 5 shows a top slice started and mat formed. Fig. 6 shows ex-

tremities of orebody being taken out in advance of central portion. This is done where a main extraction tunnel is immediately below the slice, in order to obviate repair costs in main levels.

It is evident to the writer that in future developments in these prop-

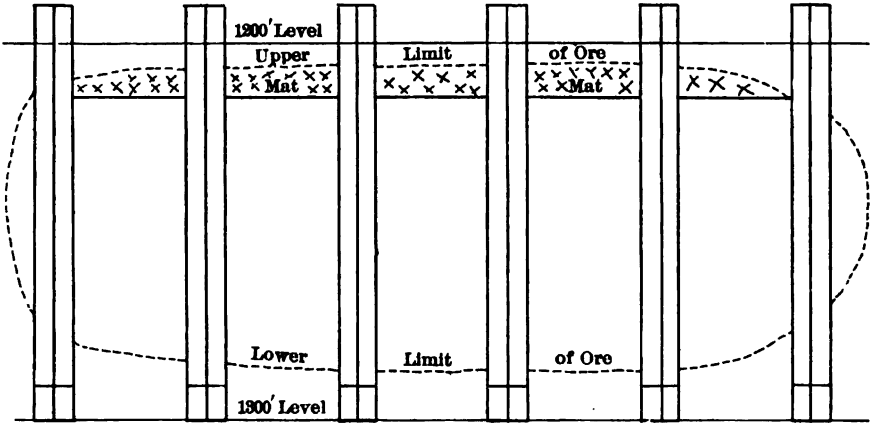


FIG. 5.—TOP SLICING.

erties, wherever orebodies are developed which are adapted either to top slicing, cut-and-fill, or shrinkage methods, these methods will be found to be much more economical than square setting, which has been

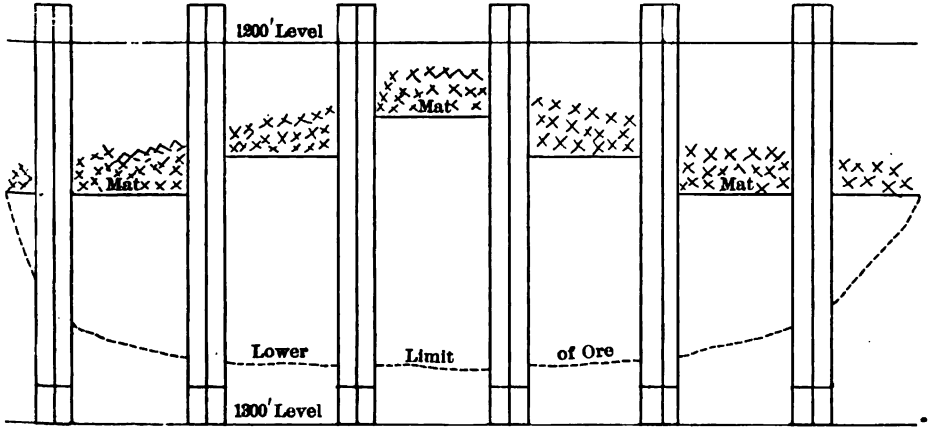


FIG. 6.—TOP SLICING.

in vogue almost exclusively in the past. It must be remembered, however, that there will always be a large proportion of the ore in these properties extracted by the square-set method, as it has undoubtedly some advantages under varied conditions that the other systems do not have.

Ventilation

Until recently, natural ventilation, aided by exhaust from drill machines, small 5-h.p. blowers, and compressed air, was the only means of ventilating the extensive workings of the entire mine. While the temperature in stopes was not very high, the relative humidity in most places exceeded 90 per cent., which made the mine air oppressive. A mechanical ventilation system was completed in the Gardner during August, 1913. The results thus far attained in improved working conditions and increasing the efficiency of the men have justified the installation of similar systems in the Lowell and Sacramento divisions.

The pressure system of ventilation has been adopted in the Gardner. Two Sirocco blowers, situated near the 900 station, deliver a total of 70,000 cu. ft. of air per minute. This entire volume of air is so coursed as to ventilate the workings from the 1,000 to the 600 levels, whence it exhausts through the shafts of the Calumet Arizona Mining Co.

Lowell Fire District

In the Lowell division, an old fire extends from the 1,000 to the 1,300 levels. Water is being run into this fire area, and after it penetrates the hot zones the water is charged with copper sulphate. On the 1,300 level there is a concrete precipitating plant, which is 500 ft. long and 4 ft. wide. The acid waters percolate among tin cans and scrap iron, and deposit their copper content.

In order to have drifts and raises that are secure for the purpose of conducting gases which come from the fire district, those that are most important have been heavily lined with concrete.

Concrete Pockets and Raises

It has been found very economical for certain kinds of ore to put in concrete pockets and cylindrical raises in storage chutes, as the upkeep cost is practically nothing, whereas, the maintenance of timber in storage chutes is expensive.

Fig. 7 shows a sketch of a concrete pocket built particularly to handle sticky ores from the Dividend slice.

Figs. 8 and 9 show a concrete storage pocket at the Sacramento hoisting shaft designed to handle wet aluminous ores.

Copper Queen

An interesting feature of the mine is that, at present, operations are being conducted through large areas of old stopes, and ore which was regarded as waste in former years can now be mined at a profit. A large amount of this work is being done in the Czar and Holbrook divisions.

It may also be stated that the mine, during 1913, has produced 867,481 tons of ore, yielding 97,181,725 lb. of copper; 15,573 tons of lead ore have produced 5,701,628 lb. of lead.

About 104,000 ft. of development work was done for the year, of

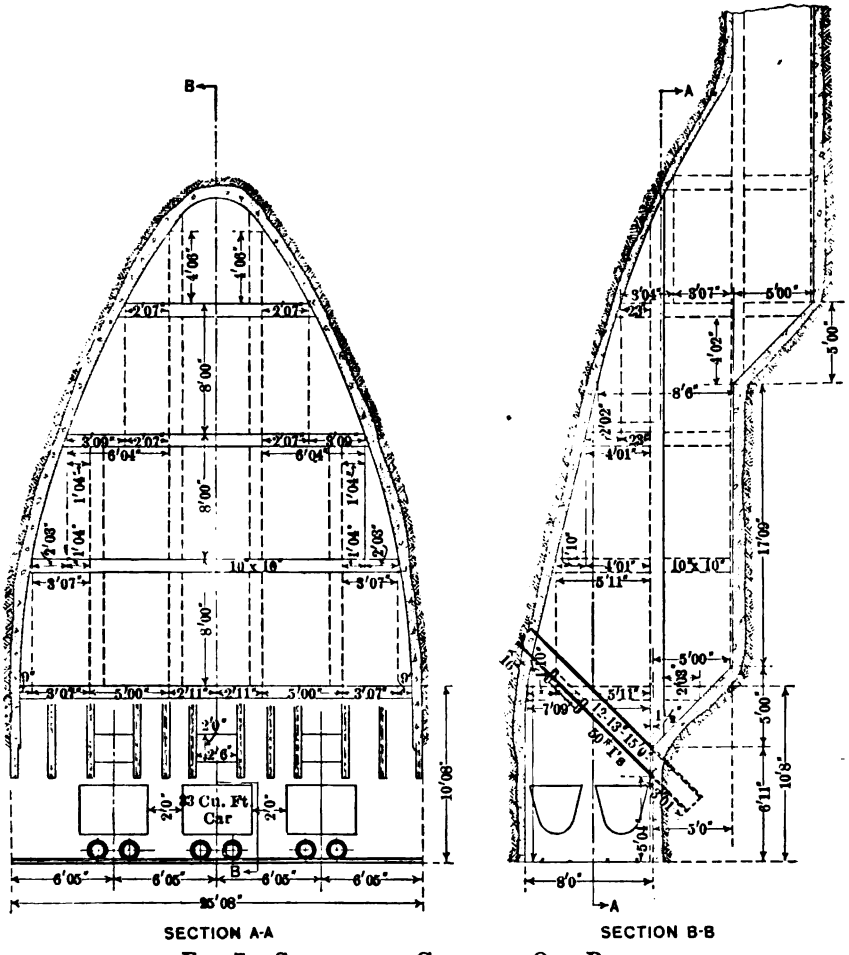


FIG. 7.—SECTIONS OF CONCRETE ORE POCKET.

which 70,000 ft. was done on contract. The amount of timber used for the year was 18,645,713 ft.

The mine has produced a total of 1,176,718,905 lb. of copper to January, 1914.

The Copper Queen, which is often referred to as a mine, constitutes a group of mines that are operated by the Copper Queen Consolidated Mining Co.

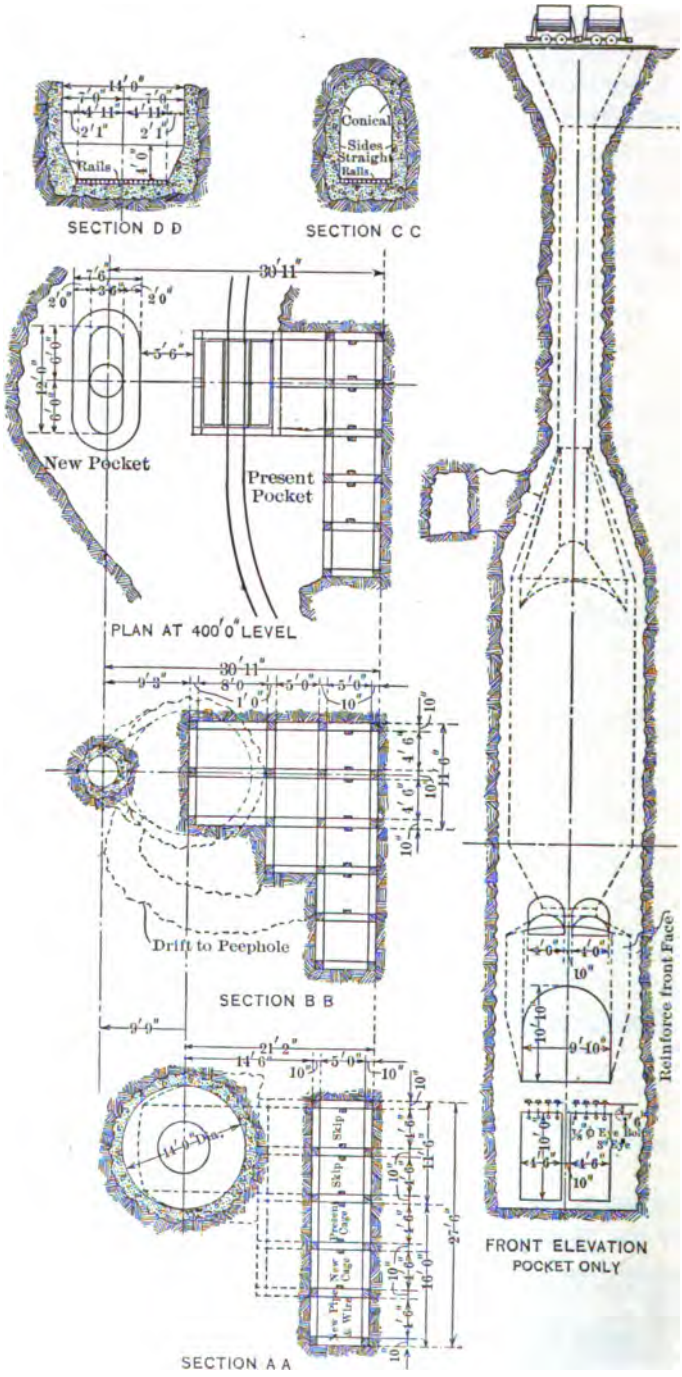


FIG. 8.—PLAN, ELEVATION, AND SECTIONS OF 260-TON CONCRETE ORE POCKET.

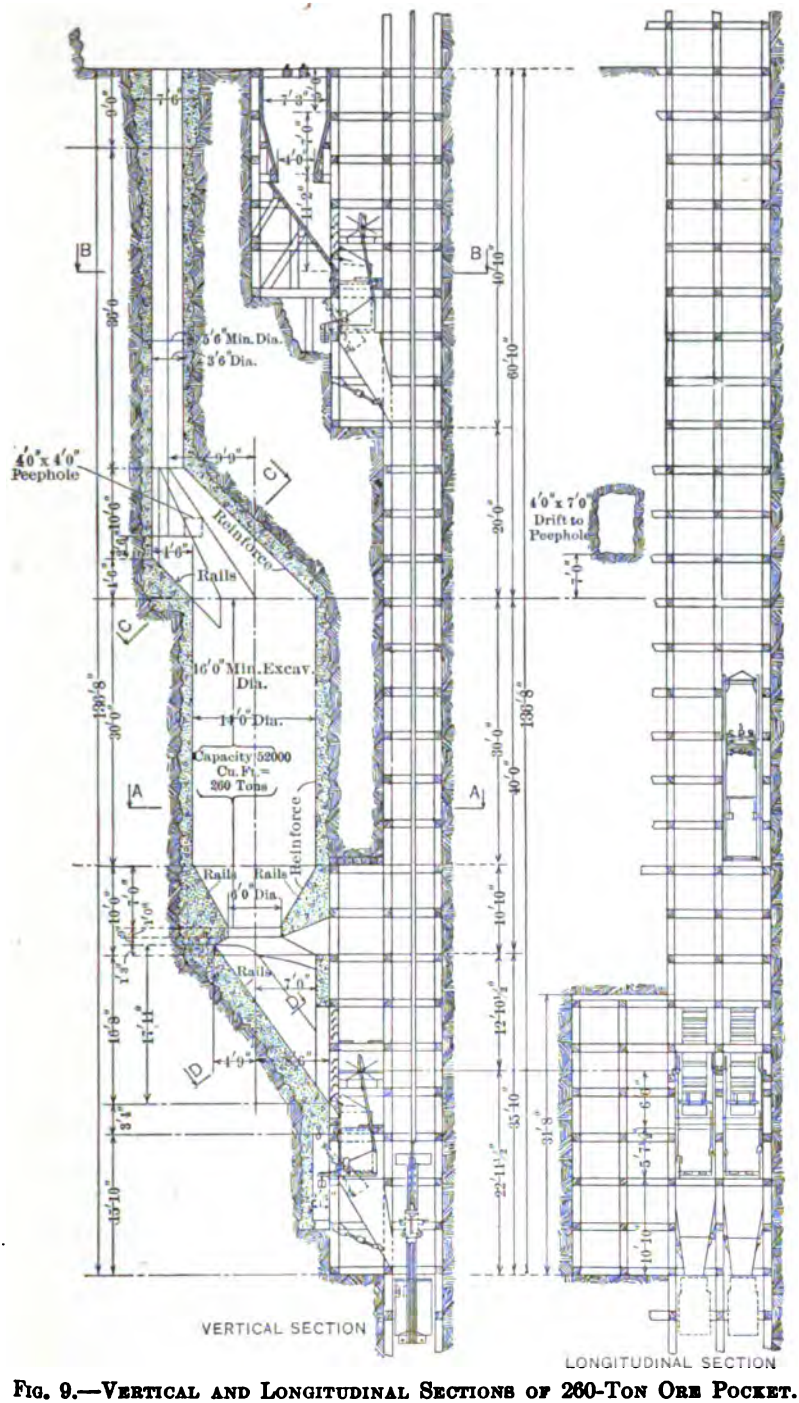


FIG. 9.—VERTICAL AND LONGITUDINAL SECTIONS OF 260-TON ORE POCKET.

Draining Kerr Lake

BY ROBERT LIVERMORE, COBALT, ONT., CANADA

(Salt Lake Meeting, August, 1914)

IT has been a noteworthy feature of the Cobalt camp, that many of the valuable ore deposits have been covered, wholly or in part, by small but usually deep lakes, such as Cobalt, Cart, and Peterson lakes, and with the subject of this article, Kerr lake. In the early days of the camp many water-covered areas, which have since been found to be valuable, were ignored or neglected by owners, or simply perfunctorily staked for water rights. Kerr lake is probably the evidence of a fault or sharp fold, having a general east and west strike, and containing a series of ore-bearing fissures, parallel generally to the strike, although having some notable exceptions to the general rule.

On the north side of Kerr lake the formation is of diabase, while on the south side it is of Huronian slates and conglomerates. These Huronian measures extend out under the lake, dipping gently to the north to a lateral distance not yet definitely determined, owing to the lack of development under the deepest parts of the lake. Probably the contact between the diabase and Huronian or else between the former and the underlying Keewatin formation occurs somewhere near the center of the lake.

Kerr lake originally covered 45 acres. Of this the Kerr Lake Mining Co. owned 12, the Drummond Mine, 7, and the Crown Reserve Mining Co. the remainder of 26 acres. Since the latter's property was originally entirely under water, it was necessary for the owners to make land for buildings and shaft room. Accordingly, in 1908, a trench was blasted out which deepened the outlet, and lowered the lake 8 ft.

As many of the rich veins of the Crown Reserve and Kerr Lake companies were under water, mining was pursued under some disadvantages. Fortunately the rock is tight and solid, and mining has been done sufficiently far from the surface to avoid unnecessary risk. Nevertheless, there was always danger of encountering open seams, through which too large a flow of water for comfortable working might come. Careful soundings were made over the veins, by means of steel-shod pipe, through mud and water to bedrock, but there was sometimes an element of uncertainty as to whether actual bedrock had been reached. Furthermore, the necessity of leaving safe backs between workings and lake tied up large quantities of ore, and made development in certain

directions hazardous. The desirability of complete draining of the lake was early recognized, and several plans were proposed for its accomplishment, but owing to various difficulties, both of an engineering nature, and of securing agreement between the various companies affected, did not come to fruition.

Crown Reserve and Kerr Lake Companies Undertake Dewatering of Lake

During the summer of 1912, the Crown Reserve and Kerr Lake companies gave serious thought to the subject, and came to the conclusion that if the dewatering were to be done, it could best be accomplished by pumping rather than by an earlier plan of tunneling, both on the score of simplicity and expense. Preliminary surveys were made over the route the water was to follow, viz.—through the Kerr lake outlet, to Glen and Giroux lakes, and thence by the outlet stream of the latter to the Montreal river, a total distance of 8 miles.

In April, 1913, an act of Parliament permitted the draining of such bodies of water as were an obstruction to mining operations, which act removed one of the hindrances hitherto existing to this and similar plans; and in May, 1913, the purchase of the 7 acres owned by the Drummond interests, under Kerr lake, removed the last vital objection to the undertaking. Permission was granted by the mining commissioner in May, 1913, and work was begun at once.

Kerr lake at this time covered an area of 30.35 acres, having been reduced from the original area of 45 acres by the work of the Crown Reserve Co., above mentioned, and by filling in by the waste dumps of the two companies. Of the total area 18.5 acres belonged to the Crown Reserve, 6.54 to Kerr Lake, and 5.31 acres formerly of the Drummond Mine, to the two first-named companies jointly.

There are no inlets of importance in Kerr lake, and its one outlet carried off water running at the rate of 300 gal. per minute in the freshet season to nothing in the dry season. It was not thought that the lake was spring fed to any extent, but that it maintained its level simply from rain and melting snow. Soundings had established its greatest depth at 100 ft., nearly 20 of which was soft mud. It was estimated that the lake contained 400,000,000 gal. of water and liquid mud in all. In any plan for dewatering, the mud had to be taken into account, as a large deposit of th's left behind would leave parts of the exposed surface in as bad shape as ever for mining purposes.

Pipe Line Direct from Kerr to Giroux Lake

It had been planned in the first surveys to pump the water through the old outlet, whence it would run through natural channels by Glen and

Giroux lakes to the Montreal river, but while this would have been a simple and feasible plan for the water only, the problem involved by the proper disposal of the mud prevented its adoption. Hence the final surveys were run in a direct line from Kerr to Giroux lake. A pipe was to take the water over this line, crossing the Kerr Lake property, the township highway, the tracks of the Temiskaming & Northern Ontario Railway, and several rights of way of power and compressed-air lines, before entering Giroux lake. The greatest elevation of the line above Kerr lake was 53 ft., and the linear distance from lake to lake, 2,400. ft. The difference in elevation between the two lakes was 20 ft. Giroux lake covers about 230 acres, and is of great depth, with an ample outlet, so that the disadvantages and objections of possible blocking of channels and flooding of other properties met with in the original plan, were done away with.

The problem which then presented itself was the installation of a pumping plant capable of handling both water and an indeterminate amount of solids against a static head varying from 53 ft. at the start to 153 ft. at the end of operations, when the lake should have been completely drained, through a pipe line large enough to eliminate the friction factor as much as possible, yet small enough to give sufficient velocity to mud-laden water. Allowing an ample friction factor, it was estimated that the total static and dynamic head would approximate 185 ft. The pumping plant, further, must have a variable base since the shore line would be constantly changing, and the pumps must have a great range of action to maintain an even flow of liquid of changing density against an ever increasing head.

Pumps Placed on Scow

On account of the steep and irregular shore and bottom of the lake, and the cumbersome nature of the machinery, a plant mounted on a movable base ashore presented obvious disadvantages, so it was decided to place the pumps upon a scow. This scow was to be kept near the shore because if anchored in mid-lake a good many pontoons would have been necessary to support the heavy pipe; also, since flexibility of the pipe line at some point was essential, on account of the changing level, it would have been difficult to anchor both scow and pontoons so as to be flexible enough and yet not too susceptible to wind and movement of the waters. Further, constant changing of mooring cables at many points would have been necessary as the level of the lake changed.

Since the Kerr Lake, Crown Reserve, and the neighboring Drummond (now Cobalt Comet) mines all took their water supply from Kerr lake, the plans included the erection of an auxiliary water-supply pumping plant on Giroux lake, large enough to fill all needs of the companies concerned.

Machinery and accessories were ordered in May, 1913, and with the receipt of the first shipments of pipe during the latter part of that month active work was begun.

FIG. 1.—SCOW IN POSITION, READY TO PUMP.

The scow on which the pumps were to be mounted was built from designs of the Kerr Lake Mining Co.'s engineer, by the Crown Reserve

FIG. 2.—LAKE LOWERED 10 FT.

Mining Co. at its shops on the shore of the lake. This scow (Figs. 1 and 2) was designed to be as compact as possible, yet to allow plenty of

working room when machinery and fittings had been installed. Stability, rigid construction, light draft, and carrying capacity up to 70 tons displacement were all essential, and were taken into account in the plans.

The hull was built of Western fir throughout, dimensions over all being 40 by 20 ft., depth of sides 4 ft., and dimensions of bottom 31 ft. 6 in., a lowing a 4 ft. 3 in. overhang at bow and stern. For the bottom, 3-in. planks laid lengthwise were used; across these other 3-in. planks were laid at 2 ft. 3 in. centers as sills for the uprights supporting the deck timbers. The boat was divided longitudinally into three sections; the sides, the center, and the two intermediate divisions of the framework were built of 6 by 6 in. timbers at top and bottom, each division spaced 4 ft. 9 in. apart. The uprights were 6 by 6 in. posts set on the bed planks at 2 ft. 3 in. centers and bolted to the longitudinal timbers. The deck beams, also 6 by 6 in., rested on the latter at 2 ft. 3 in. centers, and were bolted to them and to the supporting posts. Deck and sides were of 3-in. plank laid lengthwise, and spiked and bolted to the frame. A 2-in. flush was given to the deck, and hatches provided fore and aft for entrance to the hold. The whole boat was thoroughly calked with tar and oakum. Two anchoring spuds were placed at each side of the stern, for holding the scow firmly against the thrust of the pumps. These were 16 by 10 in. fir timbers, 30 ft. long, iron shod, and held in place by guides 14 by 10 in., which were tied by iron rods to each other above and below water. The spuds were raised and lowered by rack and pinion with spoke attachment. Besides the spuds $\frac{5}{8}$-in. wire hawsers were provided, to connect with shore at each corner of the scow. In practice it was found that on account of the shifting nature of the mud, and the difficulty of finding firm bottom for the spuds, the hawsers were more useful, and with the aid of small yacht capstans set up on the boat and on shore, were ample to hold the scow in any desired position.

After launching, which was accomplished without trouble, by jacking up the boat to the proper angle, and placing greased ways beneath, it was towed to the east end of the lake and ballasted with about 10 tons of bagged gravel. A temporary crane was rigged on deck for handling the pumps, motors, and heavier pipe and valve fittings, which were hauled by wagon from the cars, and shipped on board at this point. The pumps and motors were assembled on their base, the bed plates of each unit bolted through 8-in. sills to the deck, and the pipe and valve connections between the pumps made. The scow was then towed back to the yard for housing and final fitting. A weather-tight house of light construction was erected, to cover all of the deck except a small space at the end, where room was needed for operation of the spuds and capstans.

Four Centrifugal Pumps Used

The plant consists of four single-stage centrifugal pumps, arranged in two units (Figs. 3 and 4). Each unit comprises a compound pumping outfit of two pumps, direct connected by a flexible coupling to the motor which is placed between the two. These have 12-in. side suctions leading outboard, and 10-in. discharges facing inboard. The pump shells are 1½ in. in thickness, with removable side disks. The runners are of

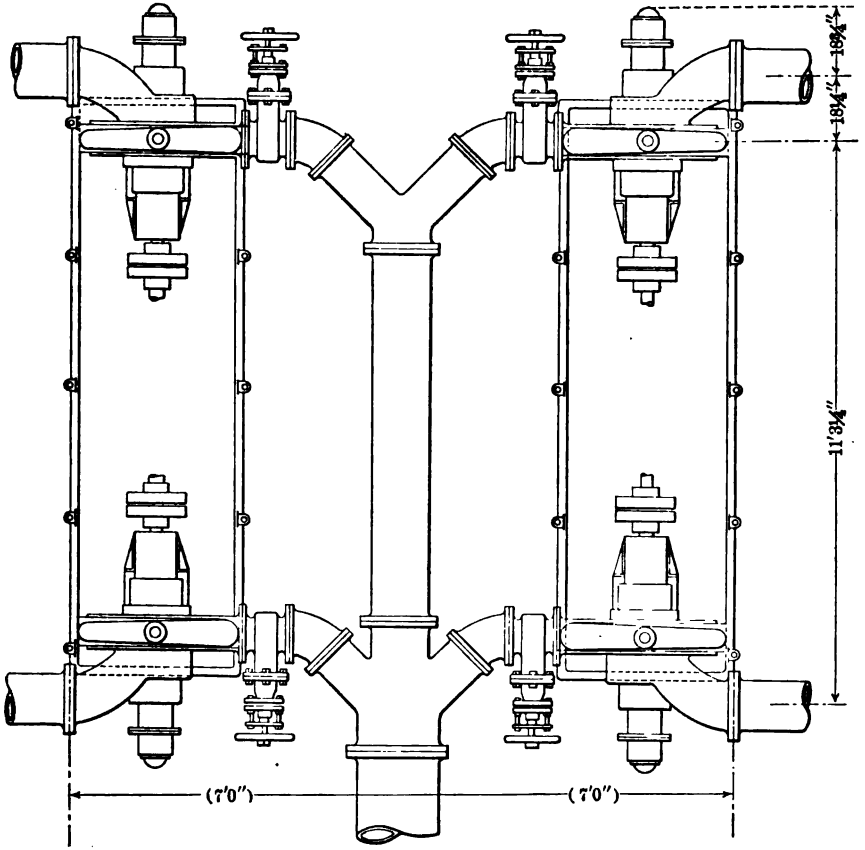


FIG. 3.—PLAN OF PIPING FOR PARALLEL OPERATION OF PUMPS.

the inclosed type, of heavy design, and capable of delivering solids up to 4 in. in diameter. The shafts are of steel extended on each side of the pumps, and carried by outboard ring oiling bearings, fitted with removable babbitt-lined shells. Adjustable thrust bearings are also provided to take possible unbalanced end thrusts.

Each unit was designed to deliver not less than 3,000 gal. per minute at the greatest elevation encountered during the operation, with a mech-

anical efficiency of not less than 60 per cent. At the start the four pumps were to work in parallel (Fig. 3), each delivering through its 10-in. discharge into a central pipe, the flow from the two sternmost pumps carrying through a length of 14-in. pipe until abreast of the forward pumps, when the flow from all four entered the main 20-in. line. All pipe connections on the scow were specially cast for the work. For the parallel connection a flanged Y joined the stern pump discharges to the 14-in.

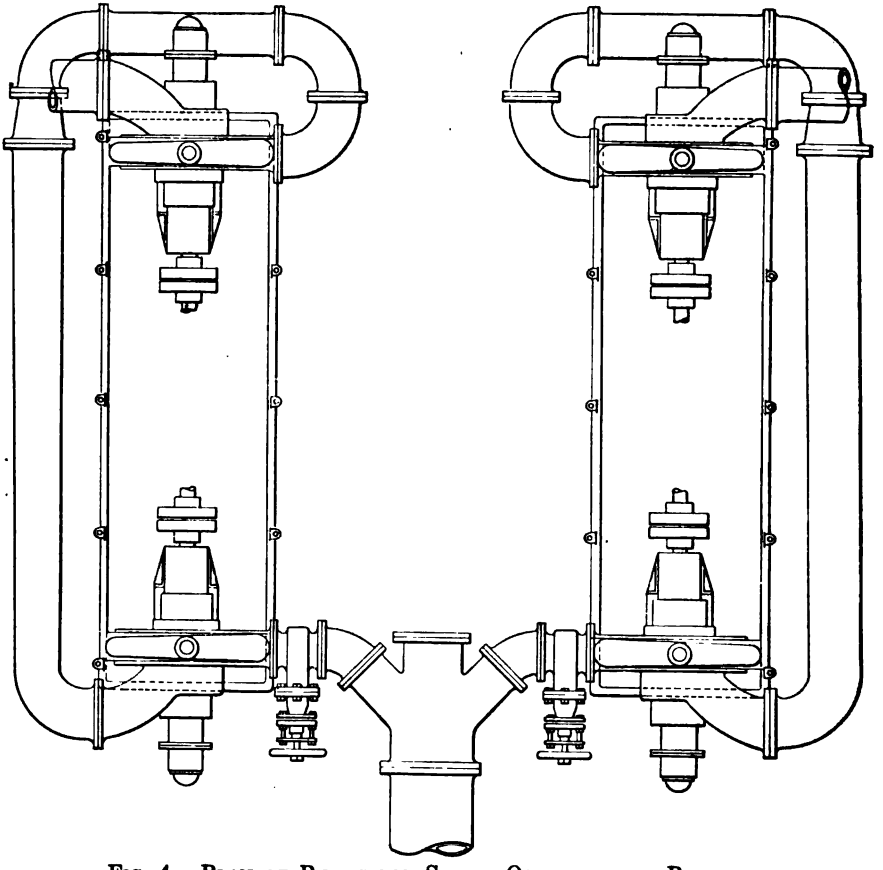


FIG. 4.—PLAN OF PIPING FOR SERIES OPERATION OF PUMPS.

pipe above mentioned, which extended along the center of the scow to another Y bored to receive it, and connecting the forward pumps to the main line. Gate valves were set in between each discharge and the main line, so that by disconnecting the motor and closing the valve any one pump could be closed off without affecting the others.

A change from parallel to series operation (Fig. 4) could be made at whatever point in the undertaking the increase of head and density of

material made it advisable. This was arranged as follows: The forward suctions, stern Y, and center length of 14-in. pipe were to be removed. Cast U-shaped lengths of pipe were provided to carry the discharge from the stern pumps outside and around to the intakes of the forward pumps, where a blank flange fitted to the 14-in. orifice in the forward Y threw the water forward to the main line. By this arrangement two units, each of a two-stage tandem pump, were obtained at some sacrifice of volume, but at a decided gain in efficiency.

For priming and keeping pressure on the impeller bearings, a 5½ by 3½ by 5 in. air-driven plunger pump was installed on the scow, which took its power from the mine compressed-air supply, and its water from Kerr lake at the start, but after its waters became too muddy, from the auxiliary supply from Giroux lake. The main suction pipes were connected to the pump shell by 90° elbows, and extended to the water's edge, where 8-ft. lengths of smooth-bore suction hose continued the intake. The suctions were fitted with flap foot valves and strainers having 3-in. apertures. The suctions were arranged on loose threads to act as a swivel joint, which, with the aid of a tackle, allowed the intakes to act at any desired depth within an 8-ft. radius.

The motors to drive the pumps are two in number, one for each unit, and connected to the pumps directly, as above described. They are 250-h.p. squirrel-cage synchronous induction motors, operating at 1,200 rev. per minute, with 2,200-volt, three-phase, 60-cycle current.

The power was obtained from the line of the Northern Ontario Light & Power Co., which passes through the property. A 2,200-volt line was carried without transforming, to a switch house built for the purpose, on shore near the scow. Here, oil and knife switches, meters, etc., were installed, whence heavily insulated copper cables were led to the scow. The connections on the scow were made through oil-switch, panel-mounted knife switches, and starting compensators for each motor. Lightning arresters were placed outside the deck-house. Cables inside the house were carried in pipe, and below deck where possible, to the various connections.

Main Line of Spiral Riveted Pipe

A 20-in., 14 gauge, spiral riveted pipe was used for the main drainage line. Bolted steel joints were used, except for a few flanged lengths on angles, and between the scow and shore. This type of joint was used on account of its great flexibility and capacity for taking up expansion and contraction. These qualities were very desirable here, where extreme changes of temperature occur, and where portions of the line, both at the feed and discharge ends, had to be shifted from time to time. This pipe was supplied in 32-ft. lengths, weighing 1,500 lb. to the length. The

pipe was laid either on the ground or on simple bents, two to each length, keeping it in as straight a line, and as free from hollows as possible. What few angles there were, were calculated in the survey, and were met by specially cast flanged elbows, to which lengths of pipe flanged at one

FIG. 5.—CROSSING OF 20-IN. DRAINAGE LINE OVER T. & N. O. RY. TRACKS.

end were bolted. A little ditch work, and rock drilling and blasting were necessary, especially where the line crossed under the main highway by culvert, but in the main, inequalities of the ground were made up by the bents. The chief engineering difficulty met with was in carrying the

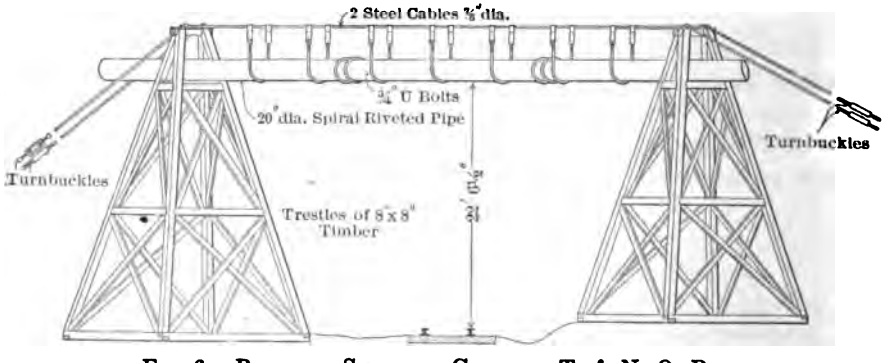


FIG. 6.—PIPE AND SUPPORTS CROSSING T. & N. O. RY.

pipe over the tracks of the Temiskaming & Northern Ontario Ry. at the minimum height allowed above rail, of 22 ft. 6 in., an action made necessary to avoid causing a hollow in the line if carried under the tracks.

At this point (Figs. 5 and 6) two wooden towers of the ordinary tram

type, built of 8 by 8 in. posts on a 16-ft. square base, well tied and cross braced, were erected on each side of the track. The angle of crossing made a span from one support to the other of 90 ft. Saddles for the pipe were provided 4 ft. below the top of the towers. Over the tops, which were iron-shod cross pieces, two 7/8-in. steel-wire cables were passed, 20 in. apart, and carried to ground. These were anchored by "dead-men" made of stout logs sunk to a depth of 6 ft., and weighted with stone. Turnbuckles were provided, by which the cables were stretched as taut as possible.

The pipe was then laid on the bents and joined as far as the first tower. Three flanged lengths were then laid alongside on the bents and bolted together to make the span. One end of the joined lengths was slung in a carrier, made of an iron loop ending in sheaves to run on the two cables, and pushed out over the railway to the far tower. Connection was then made with the pipe already laid. The span of pipe across the railway was supported by 2 by 1/2 in. strap-iron hangers passing under the pipe and hooking over the cables at 8-ft. intervals. Each hanger had a turn screw and nut, to take out all sag in the pipe. An angle flange and two connecting lengths carried the line from the tower to a rock-filled crib pier, where another angle flange was connected, and anchored by long eyebolts passing to a "dead-man" to guard against displacement of the pipe by the force of the water coming over the sudden drop from the tower. The continuing line was carried on bents as before across a swamp and down the final incline to Giroux lake.

Flexibility of connection between the floating scow and the fixed pipe on shore was secured as follows: The Kerr lake end of the pipe (see Figs. 1 and 2), which reached shore at an angle of 15°, was anchored by strap and bolts set in a cement pier at water line. The end of the pipe facing the water was flanged, and to this was bolted a 20-in. flexible ball joint having a maximum swing of 27°. To this joint in turn two expansion joints giving a lateral play of 16 in. were bolted. At the start, one 32-ft. length of flanged pipe was connected to these, and to a similar ball joint coupled inboard on the scow at the main discharge. A 12-in. bypass, with a gate valve, was inserted on the flanged length, for draining the line.

Water Supply Pumped from Giroux Lake

The water supply for the different properties formerly taking water from Kerr lake was obtained from Giroux lake, as already mentioned. It was at first planned to mount the pumps on the shore of Giroux lake, and carry the suction intakes out on piers, but the difficulty of placing piers firmly on the steep, smooth bottom, and of protecting the suction pipes against freezing, made a change of plan advisable. A well, 10 ft, square by 12 ft. deep, was sunk by air drills in the solid rock on shore

almost at water line, leaving a thin shell between lake and well. When the well was completed, a series of flat holes drilled at a depth of 5 ft. below water evel and "looking out" under the lake, was put in. The holes were heavily loaded, blasted simultaneously, and a clear connection made between well and lake. A 14 by 20 ft. pump house (Fig. 7) was erected on cement foundations over the well, in which 10 by 10 in. timbers were set at 3-ft. intervals, as sills for the pumps and equipment.

The pumps for this water supply are two in number, of the single-stage turbine type, each capable of delivering 500 imperial gallons per minute against a total head of 175 ft. The speed of operation is 1,750 rev. per minute. The pumps are mounted on the same base plate with a 45-h.p., 2,200-volt, three-phase, 60-cycle motor, directly connected to the pumps with flexible pin coupling. The equipment comprises auto-starters, oil and knife switches, lightning arresters, and meters.

FIG. 7.—DISCHARGE OF 20-IN. DRAINAGE LINE, AND WATER SUPPLY PUMP HOUSE, GIROUX LAKE.

Water is pumped through an 8-in. spiral riveted bolted-joint pipe 1,300 ft. in length, against a 125-ft. head, to a 46,000-gal. receiving tank erected on an eminence between the two properties. From this tank the supply for the different mines is delivered in 6-in. wrought-iron pipe, by gravity.

Owing to the severe cold of the winters, great care was necessary in protecting all water lines, and the methods used merit some description. The main 8-in. line was inclosed in a 4-ft. plank box, filled with sawdust and covered on top with paroid. A 1-in. steam line from each of the two mine heating plants was run alongside the pipe to the pump house, where it was run through a coil for heating purposes, and thence to the well

inlet to keep the intakes free of ice. As muddy water was to enter Giroux lake from the drainage operations, a filter dam made of jute, supported on spiles, was placed around the inlet, which formed a small pond of comparatively clear water. Two gate filters of 8-oz. duck were placed over the direct inlet, further to clarify the water.

A signal system (Fig. 8) from tank to pump house was installed, in order to keep the water in the receiving tank at a fairly constant level. This consisted of a semicircular wooden disk revolving on bearings placed in the center of the curve, and supported on brackets fixed to the inside wall of the tank near the top. The disk was equipped with a 10-ft. arm and 10-in. cylindrical float. On the edge of the disk were

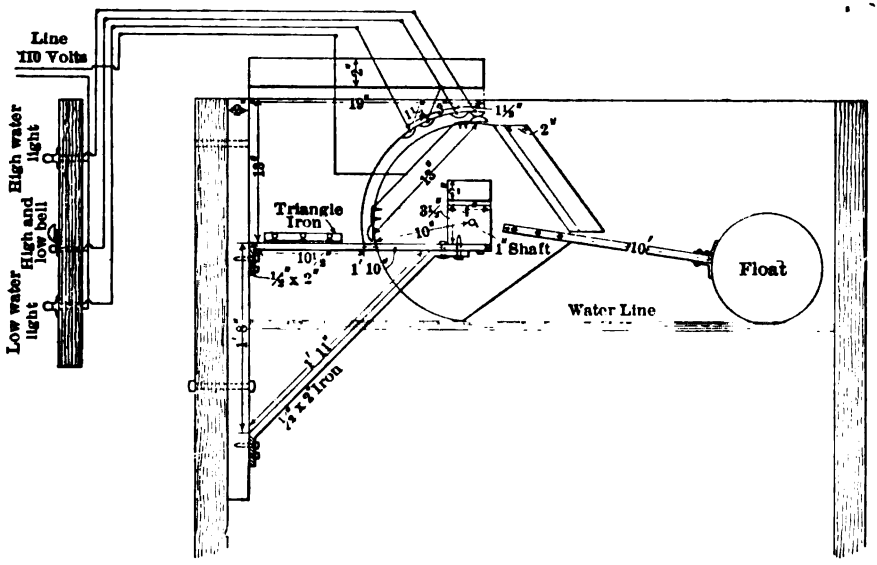


FIG. 8.—DETAILS OF ELECTRIC SIGNAL CONNECTIONS.

placed two electrical contact points one quarter of the circumference apart. Facing the disk edge was placed a stationary block, having a concave surface, matching, and clear of, the disk edge. On the upper part of this surface were placed two pairs of electrical contacts. Two incandescent lamps and an alarm bell were placed in the pump house, arranged on a panel so that the bell had a light above and below. Electrical connection was made by wire from a 110-volt circuit to the two moving contacts on the disk. Circuit was made between the inside contact of each pair on the stationary arm and the alarm bell, and between the outside contacts and the lights. The operation of the signals was now as follows: When the water had lowered sufficiently, the lower moving contact made connection with the lowest fixed contact, and lighted

the warning lamp on the panel, showing "near low water." The water continuing to lower, shortly afterward contact was made with the lower of the inside fixed contacts, and the alarm bell was rung. In the same way, when the water in the tank was nearing high mark, the lamp marked "near high" was lighted and then the bell was rung, only in this case the upper moving contact and pair of fixed contacts were in play. The contacts were spaced on the disk and concave surface so as to obtain these effects at the proper stages.

Drainage Operations

The Kerr lake pumps were started on Aug. 28, 1912. Some difficulties of a temporary nature were met with at the start, but there were, on the whole, few delays or hitches in the operation. It was found that too sudden stopping of the pumps caused vacuums to form which made one or two lengths of pipe show a tendency to collapse, but this was remedied by the insertion of check valves at threatened points. A tendency of the pipe to sag after receiving the full weight of water was observed on the span over the railway, and the structure was strengthened by the addition of two more $\frac{7}{8}$-in. cables stretched and anchored in the same way as the original ones, and by the construction of arm props on each tower, hung out over the track by $\frac{3}{4}$-in. rods, and set in notches on the legs, which shortened the unsupported span of pipe by 24 ft.

The water was lowered steadily until the depth under the scow became too shallow for convenience, when a new flanged length of pipe was inserted and the scow moved out a corresponding distance. The extra length was supported by trestle bents having an adjustable block and tackle sling to allow for the falling level of the pipe. The operation was repeated as often as it became necessary to make a move until the suspended line became too cumbersome, when the shore ball joint was moved out to a new pier and the connecting pipe shortened accordingly.

It had been intended to stir up the mud as much as possible by agitation while there was plenty of water in the lake, so that the mixture flowing through the line might be as liquid as possible. Various methods were tried, such as directing a stream, pumped from the lake by an auxiliary plunger pump of 300 gal. capacity, through a 4-in. hose with monitor attachment, into the mud, both from shore and from a small scow, but it was found impracticable, since the mud was of such consistency that although the bulk of that exposed by the lowering of the water flowed into the deeper part almost without sluicing, that under water had a glue-like tendency to stay in banked masses near the shores as long as any water remained in the lake to hold it back. Such mud as was left above the water soon dried and became compact to a short depth, so that it offered little difficulty to mining or prospecting, but that in the center of the lake constantly increased in depth with the influx from

the sides, and as rapidly diminished the clear-water area. Tests made on this mud showed that its water content was very high, nearly 80 per cent., so that it was decided to pump out the remaining water entirely, and with the suctions resting in the mud, trust to the pumps handling it with the aid of the small streams of water flowing into the basin from the mines and mills. The change of the pumps from parallel to tandem operation, provided for as above described, was made in order to obtain better efficiency with the heavier material, and after some experimenting with the proper mixture of mud and water, and with the size of the strainer openings, a fairly steady stream of liquid mud varying from 6 to 20 per cent. solids was maintained through the line.

Some trouble was had from the muddying of the water in Giroux lake near the mine supply pumps. This was met by carrying the pipe line on a curve of 45°, to which the flexible nature of the bolted joints adapted it admirably, to a wooden flume, built along shore, and extending to a remote cove of the lake, after which no more trouble was experienced.

At the time of the first proposals for the draining it had been feared that the health of the people in the rather thickly settled vicinity of Kerr lake might suffer, but fortunately these fears had no justification; in fact if anything the reverse was the case, since the more than doubtful water supply of Kerr lake was replaced by the purer water of Giroux. At one period when most of the clear water had been pumped away, some embarrassment was caused by the large number of fish which had been smothered by the mud and came to the surface. Great numbers ranging in size from small perch to eels and pike 30 in. long lay everywhere, and even clogged the suctions and entered the valves. Fortunately, the situation was well taken care of by the gathering of great numbers of scavengering gulls who soon disposed of the fish, to the mutual advantage of themselves and the operators.

The pumps were run through September and October, and at intervals during the month of November, but during the latter month the increasing cold made operation difficult because of the freezing of valves and of the surface of the mud, so that work was stopped for the season the last of the month. The operations to this point were successful from the standpoint of the companies concerned. The water was removed from the greater part of the important reserves of the mines, thus leaving them free for stoping and development. About 325,000,000 gal. of mud and water were pumped, at an average of 6,000 gal. per minute for 38 actual working days.

It was demonstrated that the liquid mud lying in the deeper parts of the lake could be pumped, and that eventually the ground underlying these deeper parts could be prospected and mined at will after the removal of the remaining mud in the following open season.

Several important veins were disclosed by the draining, some of which had not been encountered in the underground workings, and some of which, although previously known and followed, were not known to be so valuable as their surface appearance later showed them to be.

One of the most important questions decided was that the long immersion of the veins had had no ill effect on their value, as exposure showed that for the most part they had suffered no oxidation or leaching of valuable minerals. In one curious instance the glacial striations in a native silver outcrop appeared as burnished and bright as if made yesterday.

From an engineering standpoint there were few miscalculations or setbacks to the work, and allowing for the usual delays in transportation and delivery of ordered equipment, the work of installation was expeditious.

The managements of the two companies, under whom the work was conducted jointly, owe much credit to their respective staffs and other employees, to whose harmonious and efficient co-operation, and many ingenious ideas in perfecting details, the assured success of a somewhat novel undertaking is largely due.

Rope Idlers in the Raven Shaft

BY GEORGE A. PACKARD, BOSTON, MASS.

(Salt Lake Meeting, August, 1914)

THE shaft of the Raven mine, at Butte, Mont., is an incline 1,700 ft. in length and dipping at various angles. At the top the dip is 70° from the horizontal, but this is gradually flattened until at the 300-ft. level the inclination is only 47°. This angle continues to the 1,100-ft. level, below which it curves with a 125-ft. radius to 78°. In addition to these variations in dip, the shaft does not lie in the same vertical plane, with the result that the hoisting rope not only rubs at intervals on both the hanging and foot walls, but presses strongly against the west dividers near the collar, while 300 ft. below it runs close to the east end plates.

The early operators used no idlers, and wall plates cut 6 in. deep by the rope resulted. Later operators first attempted to overcome the excessive friction, and the wear of rope and wall plates, by introducing solid cast-iron idlers, 3 in. in diameter. To allow for the travel of the rope from side to side some of these had to be 3 ft. long and were very heavy. Judging from the appearance of the old idlers of this type found at the mine, they often failed to turn in the bearings, which is not surprising when it is considered that they would have to make 1,000 rev. a minute under ordinary hoisting conditions.

The next rolls were made of wood, 6 in. in diameter, with an iron band about each end, and a pintle of 1-in. round steel driven in at the ends to serve as a shaft. These wore rapidly, and were soon replaced by rolls made from water pipe, 5 or 6 in. in diameter, cut to the desired length and fitted with a wooden cylinder into which the pintle was driven. Where the idlers were used on the hanging wall of the shaft the original bearing was simply a piece of $\frac{1}{2}$ by $1\frac{1}{2}$ in. strap iron, 10 in. long, turned up at the end in a circle $1\frac{1}{8}$ in. in diameter to receive the shaft. A small hole served for oiling, and common black oil was generally used, although filtered oil from engine bearings and compressor bearings was also used. When the rolls were to be placed on the foot wall the bearings were made from two pieces of 1 by 3 in. steel, 6 in. long. A half circle was cut on the flat side of each piece and the two half circles together formed a bearing. Oil holes were provided, and in some cases holes were bored through the two pieces so that they could be screwed or spiked to the wall plates.

The later practice was to forge the bearings from 1 by 3 in. steel, and to drill two holes at each end for ¾-in. lag screws, by which the bearings were fastened to the timbers. These bearings were finally used on both foot and hanging wall. Similar idlers were so placed as to protect the dividers and end plates. The difficulty of proper oiling presented the greatest obstacle to satisfactory results from this type of idler. As the clearance between the skip and the hanging-wall plates was sometimes less than an inch, there was not room for large oil or grease cups. In addition, the bearings were liable to get full of grit, especially when wet ore was being hoisted. Grease cups were generally unsatisfactory, although several kinds of grease were tried and especial attention was paid to having that

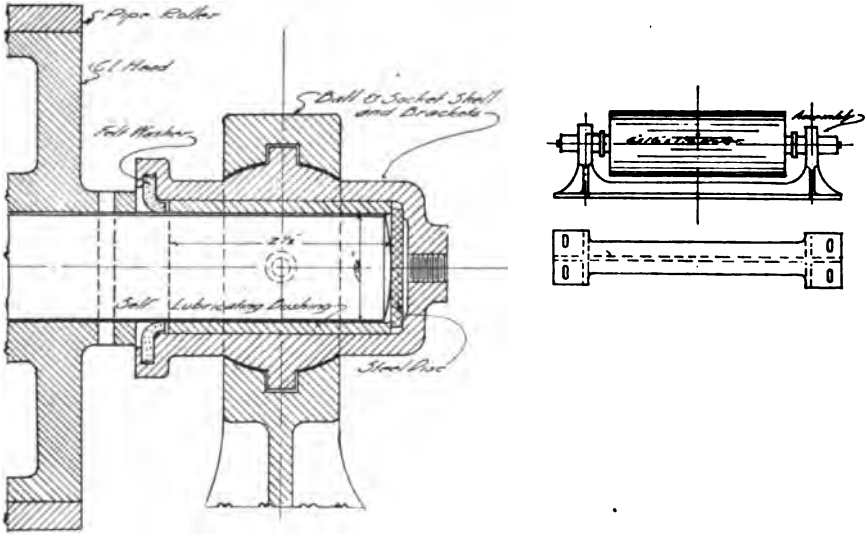


FIG. 1.—ROPE IDLER USED IN THE RAVEN MINE, BUTTE, MONT.

which was suited to the temperature of the shaft. In any event it was necessary that the rolls be examined and the oil cups filled every two days, which meant the cessation of hoisting for 2 hr. The bearings wore rapidly and the rollers tended to get out of line. The full skip weighed over 3 tons, and where the shaft flattened near the surface the pressure against the idlers was heavy. It was only by distributing this weight over idlers placed but 5 ft. apart that anything approaching satisfactory service could be obtained at this point.

To obviate the necessity for so much attention, an idler was devised, following my suggestions, by the Robins Conveying Belt Co. This idler is shown in Fig. 1. The roller is extra heavy 6-in. pipe, ⅜ in. thick, 20 in. long, in each end of which is pressed a cast-iron head, and through which passes a 1¼-in. steel shaft. This turns in a self-lubricating

bearing carried by a bracket in a ball-and-socket shell, which prevents cramping. As a preliminary to adopting these bearings two types of graphite and bronze self-lubricating bushings were tried side by side in the incline for three months. The one proving most satisfactory had cylindrical bodies of graphite ¼ in. in diameter set in the bronze, or "metalline," bushing at about ¾-in. centers. One end of the bearing is entirely closed, the end thrust being taken by a steel disk, which also serves for forcing out the bushing when it is worn. The other end of the bearing is protected from grit by a felt washer. This, however, also retains the fine particles of metal and graphite, and in time this gummy matter causes the bearings to bind. Occasional cleaning of the bushings with kerosene obviates this trouble. The cap is hinged at one side and fastened at the other with a hinged bolt so the roller and bushing can be easily removed. The bearings can be turned through 90°, and the roll turned end for end, permitting the advantage of full wear. The whole is carried in a casting which is fastened to the wall plates with ¾-in. lag screws. These idlers have been in use nearly a year and are very satisfactory. While with rapid and continuous hoisting the bearings become quite hot, they do not bind if they are cleaned occasionally.

For the lower part of the shaft, where the rope runs true and the inclination is 78°, the idlers are merely common sheave wheels, cast solid and keyed on a shaft of 1-in. cold-rolled steel. This has a total length of 13 in. These sheaves are 9 in. in diameter with a 3-in. face, having a groove 1½ in. deep, 1 in. wide at the bottom. The bearings are maple blocks 4 by 4 by 6 in., bored to receive the shaft, and provided with an oil hole. These are fastened to the wall plates with six spikes.

The wood-filled pipe idlers with forged bearings cost at Butte about $8 each, including bearings. The idler with the self-lubricating bearings costs $15, but the difference is quickly saved in decreased cost of attention. The solid cast sheaves weigh 26 lb. They cost, when fitted with a shaft, but excluding the maple bearings, $2.75 each.

Tests of Rock Drills at North Star Mine, California

BY ROBERT H. BEDFORD AND WILLIAM HAGUE, GRASS VALLEY, CAL.

(Salt Lake Meeting, August, 1914)

THE objects of this paper are:

1. To give cost data on drilling obtained at the North Star mine in California;

2. To describe the methods of testing drills employed there, giving the results of shop tests, and the correlated data obtained underground;

3. To demonstrate that these tests form a satisfactory basis upon which to judge rock drills.

The rock at this mine is either close-grained diabase in the upper levels, or tough grano-diorite in the lower levels. The vein, which has an average dip of 23°, consists of about 5 ft. of "formation" lying between walls of unaltered country rock. Of the 5 ft., solid quartz and stringers make up 18 in. of pay ore. The formation, which consists of but slightly altered country rock, is almost as hard as the unaltered walls. From 4 to 4½ ft. is sent to the mill. One-third of the holes are drilled in the quartz; two-thirds in the formation. The holes are about 4½ ft. deep and break on the average of 1.15 tons per hole. The average number of holes per stope drill shift is at present 5.65. The number of drill shifts throughout the mine in 1913 was 18,679, and the cost of labor for drilling, power, supplies, upkeep of machines and air lines, tool sharpening, and distribution, amounted to 33 per cent. of total mining expense. To do this work there are in commission 43 No. 12A, three No. 17V and eight No. 16V Waugh stopers, three Jackhamers, two Waugh pluggers, and 17 No. 8 and two No. 7 water Leyners.

For the past three years a record has been kept of the repairs on each machine and number of shifts operated by it. The figures given in Tables I and II for the cost of repairs per drill shift have been taken from this record. The underground air pressure is about 90 lb. The air consumption per drill shift was measured by aerometers. The leakage of pipe lines between compressor and meter is not included. From the number of pieces of steel sharpened for each type of drill during the year the average used per drill shift has been ascertained. For other supplies the figures given represent a year's average.

TABLE I.—*Cost per Drill Shift for Water Leyner No. 8*

Item	Description of Unit	No. of Units Used per Drill Shift	Price per Unit	Cost for Item per Drill Shift
Labor of drilling	8-hr. shift	1	$3.25	$3.25
Maintenance,				
Labor				0.10
Supplies				0.62
Power	1,000 cu. ft. of free air compressed to 100 lb.	15	0.0275	0.41
Supplies,				
Lubricants	Quarts of "Red Engine"	0.66	0.07	0.05
Hose	Feet of 1 in. 5-ply wire-wound	0.12	0.33	0.04
	¾-in. 5-ply wire-wound	0.12	0.19	0.025
Drill steel	Pieces used	16.75		
Labor sharpening and repairs				1.12
Labor distribution				0.35
Steel consumed	Pounds	2.66	0.125	0.33
Power for sharpener	1,000 cu. ft. free air compressed to 100 lb.	5.00	0.0275	0.15
Oil for forge	Bbl. of 42 gal	0.12	1.80	0.21
Upkeep of air pipe				0.09
Total				$6.75

TABLE II.—*Cost per Drill Shift for Waugh 12A*

Item	Description of Unit	No. of Units Used per Drill Shift	Price per Unit	Cost for Item per Drill Shift
Labor of drilling	8-hr. shift	1	$3.00	$3.00
Maintenance,				
Labor				0.10
Supplies				0.37
Power	1,000 cu. ft. of free air compressed to 100 lb.	17	0.0275	0.47
Supplies,				
Lubricants	Quarts "Red Engine" oil	0.33	0.07	0.0
Hose	Feet ¾-in. 5-ply wire-wound.	0.12	0.27	0.03
Drill steel	Pieces used	10		
Labor sharpening and repairs				0.30
Labor distribution				0.21
Steel consumed	Pounds	2.2	0.065	0.14
Power for sharpener	1,000 cu. ft. free air compressed to 100 lb.	1.5	0.0275	0.04
Oil for forge	Bbl. of 42 gal	0.034	1.80	0.06
Upkeep of air pipe				0.09
Total				$4.83

The 16V and the 17V Waugh drills cost 10c. less for maintenance supplies, but require 15 pieces of steel per shift instead of 10, which leaves the cost about equal.

As already stated, the drilling at this mine is one-third of the cost of delivering ore at the mill. The importance of selecting a suitable drill for the work and keeping it in its highest state of efficiency is obvious. At first, time was wasted and trouble experienced in testing underground drills which were unsuitable. Worn out drills which should have been "scrapped" were often kept in operation because they sounded all right when run against a block in the repair shop.

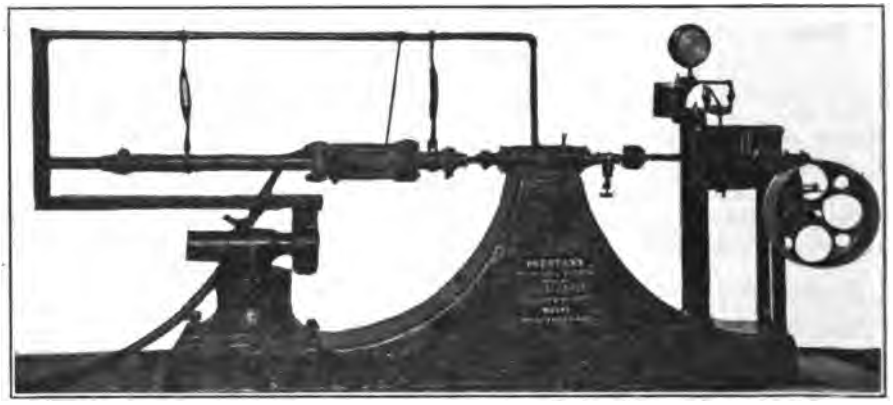
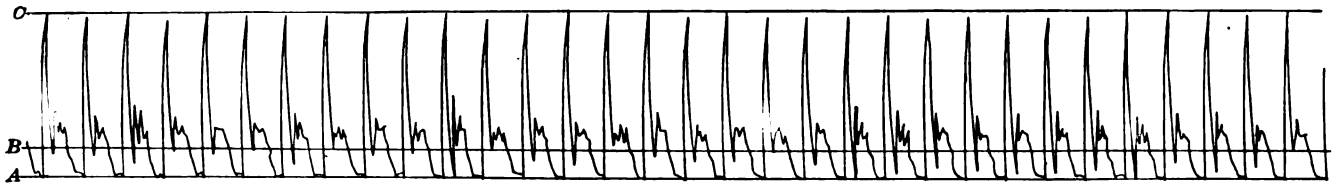


FIG. 1.—PAYNTER'S ROCK-DRILL TESTING MACHINE.

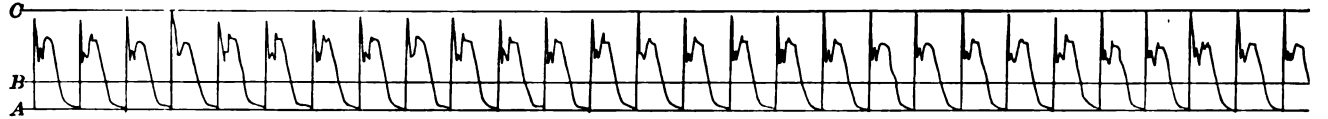
There was need for some reliable testing machine to overcome the difficulties. W. D. Paynter, who was responsible for keeping drills in repair, after experimenting for some time, perfected a testing machine, which he has patented. This machine has been described in detail elsewhere.[1] Briefly it consists of a device whereby the blow of the drill delivered against a plunger is measured by the distension of a diaphragm, oil being the medium of transmission. By means of a lever arm, the movement of the diaphragm is amplified. A pencil on the end of the lever, marking a piece of paper on a revolving drum, gives a graph of the work done by the drill over a given period of time (Fig. 1).

In Figs. 2 and 3 are given cards taken from different types of drills. *A* is the base line, line *B* represents the feed-barrel pressure in the case of an air-feed stoper. The length of the line *AC* is a measure of the energy transmitted through the drill steel to the plunger. The graph covers a period of 5 sec. The number of peaks, *C*, multiplied by 12, therefore, gives the blows per minute. To express the energy of the blow in foot-pounds, the tester was calibrated by allowing a sphere of known

[1] *Mining and Scientific Press*, vol. cvii, No. 5, p. 179 (Aug. 2, 1913); *Engineering and Mining Journal*, vol. xcvi, No. 18, p. 829 (Nov. 1, 1913).

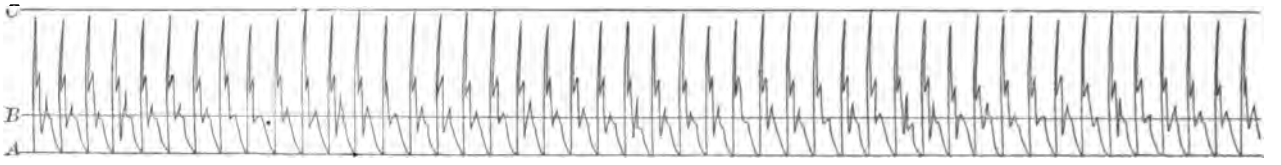


Air pressure, 95 lb.; blows per minute, 1,584; foot-pounds per blow, 55. Card No. 1.—In good condition.



Air pressure, 95 lb.; blows per minute, 1,368; foot-pounds per blow, 25. Card No. 2.—Not in good condition.

FIG. 2.—CARDS NOS. 1 AND 2.



Air pressure, 97 lb.; blows per minute, 2,352; foot-pounds per blow, 40.

FIG. 3.—CARD NO. 3.

weight, suspended as a pendulum, to fall through measured distances against the plunger. The distensions of the diaphragm corresponding to these blows were noted. By forcing oil into the system with an attached pump, the diaphragm was distended to each of these points in turn, and the static pressure corresponding to each was read on the gauge. A curve was then constructed with foot-pounds as abscissæ and pounds static pressure as ordinates. The strength of any blow, recorded on

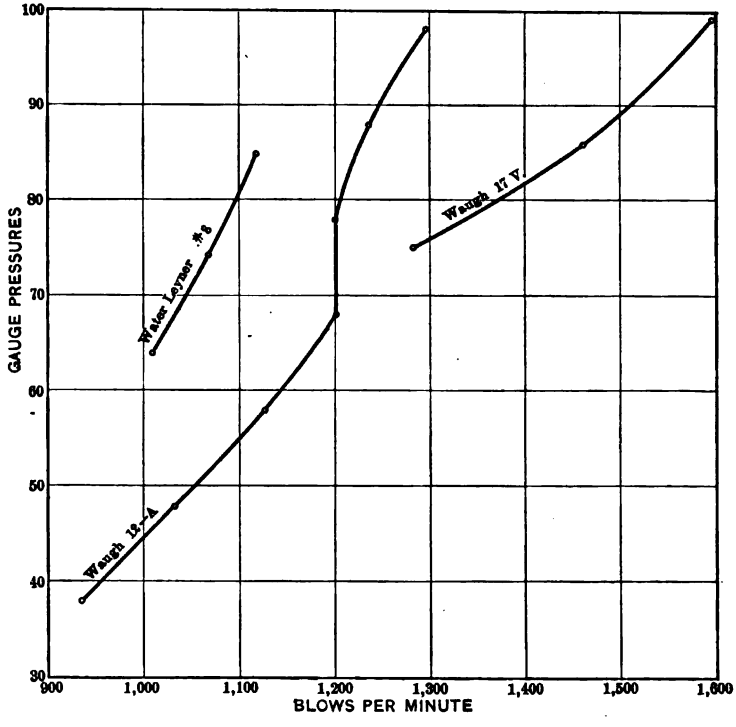


FIG. 4.—CURVE SHOWING INFLUENCE OF GAUGE PRESSURE UPON BLOWS PER MINUTE.

the graph as the line AC, may be measured by pumping oil into the system until the distension of the diaphragm moves the pencil out to point C. The pressure in the system at this point is read on the gauge. The foot-pounds corresponding to this can then be read directly from the curve. Since the areas of the plunger and of the diaphragm do not change, equal blows will produce the same pressure, even though permanent straining of the diaphragm may result in their producing unequal distensions. Errors which might be caused by these inequalities of distensions are avoided by thus expressing strength of blow in terms of equivalent static pressure instead of magnitude of distension.

The general method of testing a drill consists in first obtaining graphs in the shop and then the drilling speed underground. Having passed the first test, the drill is used in the mine, a record being kept for several months of the footage drilled. Graphs taken from time to time show whether or not the drill is deteriorating, while the card record of repairs gives the cost of its upkeep. Drills sent to the shop for repairs are not returned to the mine until they show on the tester a satisfactory graph.

From the graphs taken by testing a drill at different gauge pressures, characteristic curves may be constructed, as shown in Figs. 4 and 5.

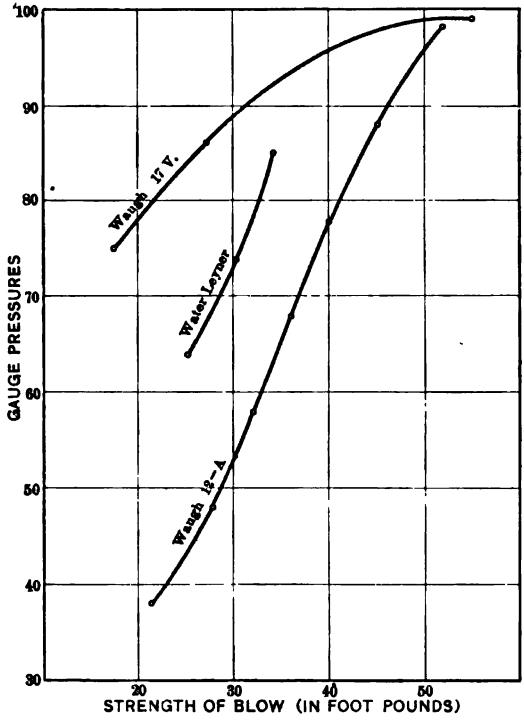


FIG. 5.—CURVE SHOWING INFLUENCE OF GAUGE PRESSURE UPON STRENGTH OF BLOW.

Below are given the results of tests made to determine, if possible, the relation existing between drilling speed, blows per minute, and foot-pounds per blow.

Table III compares (*a*) three drills striking a hard blow, (*b*) two drills striking a medium blow. These were tested in very hard ground with holes at an inclination of 25°, 1¼ in. cross steel with 2¼ and 2 in. bits being used; conditions, however, were kept the same for each test, which lasted 5 min.

TABLE III

	Foot-Pounds per Blow	Blows per Minute	Drilling Speed Feet per Minute
(*a*)	53	936	0.126
	52	1,320	0.165
	55	1,584	0.250
(*b*)	40	1,260	0.108
	42	2,352	0.195

These results indicate that drilling speed varies approximately with the blows per minute, the strength of the blow remaining constant.

Table IV compares drilling speed with varying strengths of blows. The conditions of test were: Ground only medium hard; length of each test 5 min.; 1¼-in. cross steel with 2-in. bit; holes at an inclination of 45°; drill used, air-feed stoper.

TABLE IV

	Blows per Minute	Foot-Pounds per Blow	Feet Drilled per Minute
(*c*)	1,272	48½	0.378
	1,222	43	0.447
	1,200	38½	0.308
	1,170	34½	0.250
	1,090	30½	0.188

A test under the same conditions as in Table IV, except that the inclination of the holes was 20°, gave the results shown in Table V.

TABLE V

	Blows per Minute	Foot-Pounds per Blow	Feet Drilled per Minute
(*d*)	1,272	48½	0.251
	1,222	43	0.206
	1,200	38½	0.198
	1,170	34½	0.135

A test (Table VI) was made under the following conditions: Ground hard; length of each test 5 min.; drill used, No. 8 water Leyner; 1⅛-in. hollow steel with 2¼-in. bit; holes nearly horizontal. In this test varying strengths of blow were obtained by means of stops of different lengths screwed into the ends of the valve chest.

TABLE VI

	Blows per Minute	Foot-Pounds per Blow	Feet Drilled per Minute
(*e*) (1)	1,368	40	0.162
	1,272	45	0.234
	1,260	52	0.119
	1,212	65	0.129
Repetition of test (1) for confirmation:			
(2)	1,368	40	0.135
	1,272	45	0.224
	1,260	52	0.183
	1,212	65	0.195

As already stated, Table III indicates that for blows of equal strength the drilling speed is approximately proportional to the number of blows, even when these differ as much as 40 per cent. In constructing the following curves from Tables IV, V and VI, where the maximum variation in the number of blows is 15 per cent., the drilling speed has been arbitrarily adjusted according to the results obtained from Table III, so that the effects of the strengths of blow may be comparable.

These curves all show the same tendency to flatten at above a certain strength of blow, which in this ground happens to be about 45 ft-lb. It will be noticed that the curves for the Leyner and for the stoper at 45° inclination are almost exactly similar. This is thought due to the fact

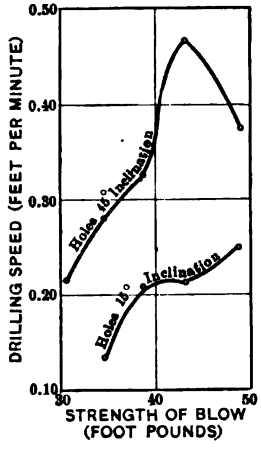


FIG. 6.—CURVES SHOWING INFLUENCE OF STRENGTH OF BLOW ON DRILLING SPEED. (AIR-FEED STOPER.) FROM TABLES IV AND V.

that in each case the face of the hole is clear of cuttings. In the cases of the flat holes, which do not clear themselves, the drilling speed is not only lower, but does not show the same peak. This suggests that where holes can be cleared of cuttings a 45 ft-lb. blow is sufficient to obtain the fastest drilling, whereas in holes where the deadening effect of cuttings exists the limit of effective blow is higher. This also emphasizes the desirability of a stoper with water fed to the face of the hole.

From these tests, and others giving similar results, it has been decided that for conditions existing at this mine, stoping drills should strike a minimum of 40 ft-lb. As none of these drills has yet been found that strikes over 55, it remains solely a question of maintenance to keep them at this point. The Leyner, with blows as high as 65 ft-lb., presents a different problem. In this type the valve was so adjusted, by means of standard plugs screwed into the ends of the valve chest, as to give at average gauge pressure blows in the neighborhood of 45 ft-lb. It is

hoped that this reduction in strength of blow will result in lessened breakage of steel, decreased repair costs, and maximum drilling speed. The adjustment has not been in use long enough to give any figures on the first two points.

The effect of setting the minimum strength of blow in the stoping drills at 40 ft-lb. has been to increase the breakage of steel by 1 lb. per

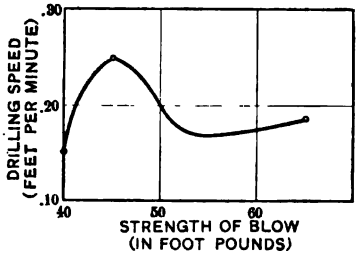


FIG. 7.—CURVE SHOWING INFLUENCE OF STRENGTH OF BLOW UPON DRILLING SPEED OF WATER LEYNER NO. 8. FROM TABLE VI.

drill shift, and the cost of repairs by 27c. per drill shift. The footage drilled, however, has been increased 15 per cent., reducing the cost per foot of hole drilled from 20.3 to 18½c. The output per drill shift has increased 10 per cent. during the same period, but this figure is complicated by the width of stope broken, and is therefore not quite comparable.

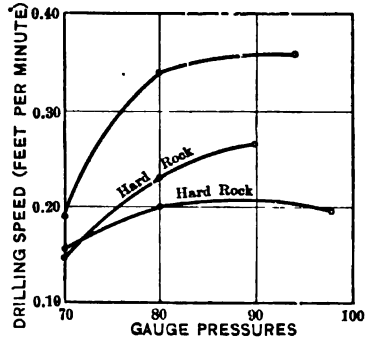


FIG. 8.—CURVE SHOWING INFLUENCE OF GAUGE PRESSURE ON DRILLING SPEED. (AIR-FEED STOPERS, WAUGH 12A.)

In the case of the Leyner drill, the cost for sharpening, breakage, and repairs makes a high cost per drill shift. Comparison with a type of machine in which these items are low is valuable, as it proves that the expense is justifiable. A 5 by 7 ft. drift in this mine requires 11 holes, aggregating 43 ft., to break a round. A 2¼-in. reciprocating drill, striking 40 to 50 ft-lb. blows, 600 to 800 times a minute, used to require two drill shifts for drilling the round, at a cost of $4.45 per drill shift, or 20½c.

per foot of hole. The Leyner does this in one drill shift for 15½c. per foot of hole.

The use of the tester in the shop has greatly facilitated repairs, as any abnormal action in the drill is disclosed at once. Experiments have demonstrated several points. Among these may be mentioned the effect of feed-barrel pressure. If the feed-barrel packing is slightly defective, the drill does not hold against the ground hard enough, causing as much as 30 per cent. decrease in drilling speed. In the 16V stopers, wear in the barrel bushing after about 250 drill shifts causes cushioning due to leakage of air, with reduction in the strength of blow, amounting to as much as 20 per cent. In the 17V type, at pressures below 85 lb. per square inch, the strength of blow drops very rapidly. Plugging some of the exhaust ports causes the strength of blow to decrease less rapidly. This makes the drill more effective at low pressures.

Examination of Table IV shows the decrease in drilling speed that may be expected with a stoper striking 30 ft-lb., as compared with 45 ft-lb. On this point it may be stated that many confirmatory tests have been made. Unless the drills are maintained at their best, the strength of blow drops rapidly (sometimes after 100 drill shifts), with resulting decrease in drilling speed. The cost per drill shift, however, remains practically the same. It is obvious what the result will be. Weston has stated in his book, *Rock Drills*, "If the whole system from compressor to drill is not maintained at the highest point of efficiency, the neglect will surely reveal itself in high mining cost."

Discussion

W. L. Saunders, New York, N. Y. (communication to the Secretary*) —Messrs. Bedford and Hague have given us a very interesting and practical paper. No better excuse for such a paper could be desired than the statement made that the expense of drilling in this mine is one-third of the cost of delivering the ore at the mill. This statement should be written in red letters, because it applies to a great many mines besides the one referred to and because it is generally true that in a mining plant, as also in papers read before the Institute, more attention by far is given to the metallurgical side of mining than to the excavation of the ore, and no study of the excavation work is properly made that does not involve a careful inquiry into the merits of the rock drill used.

Next in importance to the selection of a good rock drill, in a case where the actual drilling cost is one-third the cost of delivering the ore, is that the drill should be maintained in order and kept up to its highest efficiency. The best rock drill may run down in efficiency through accidents, wear, or other conditions which are sometimes beyond supervision

* Received Sept. 1, 1914.

or control. The machine is usually in the hands of men who know how to get work out of it, not men who know how it works. The best drill runners are not machinists, but miners. They could not tell you how the valve operates or the principle of the rotation, but they know how to get the work done. A testing machine, particularly the type described in this paper, may be of the greatest value in maintaining a uniform state of efficiency.

It must not be supposed, however, that this testing machine in itself will distinguish the value of one type of drill from another. This has been referred to in the paper, where it is shown from the record that drills developing the greatest horse power of energy were not the most efficient in drilling capacity. This is referred to on p. 353 and it is corroborated by the tables on p. 352, where the foot-pound blow of approximately 40 to 45 lb. is shown to be the most efficient for this particular class of work.

At the works of the Ingersoll-Rand Co. testing machines of practically the same principle as that described have been in use for several years. Drills are tested on these machines and afterward operated in an experimental underground mine, where it has been shown that the maximum foot-pound energy does not always mean maximum drilling capacity. The efficiency of any type of rock drill depends mainly upon the proper adjustment of the air pressure to the nature of the rock. In certain rocks a slow-moving, heavy foot-pound blow will show the best results in drilling, while in other rocks the fastest drilling is done through a rapid blow of light foot-pound energy.

The testing machine, however, is a guide in showing comparative results under similar conditions and it is a valuable means by which from time to time a drill may be tested to determine whether or not it has deteriorated. A rock drill that is not up to its maximum efficiency should be examined and the cause discovered and remedied.

It is to be regretted that Messrs. Bedford and Hague did not give us a little more information concerning the tests given in Tables I and II, on p. 347. These tables at a glance indicate that a comparison has been made between the cost of operating a water Leyner No. 8 drill and a Waugh 12A. I feel quite sure, however, that there is no comparative test intended, because, in the first place, one machine is a rock drill of the mounted type while the other is an unmounted stoper. In these tests the Leyner drill was used for heading work, which means that a larger proportion of the holes are flat and down-holes. Such work, as every one knows, is more difficult and is harder on the steel than work, commonly known as stoping, with air drills, which is usually up-hole work, the drills clearing themselves of cuttings during operation. This point has a bearing upon the statement showing the amount of steel consumed and the cost per shift for sharpening the steel. There can be no comparative value placed

upon tests of steel cost except when used in the same class of work. Furthermore, the Leyner drill was used with round steel while the stoper used the cruciform pattern. The statement is also made that 2-in. bits were used with the stopers and 2¼-in. with the Leyners.

Another point worth mentioning is that the Leyner drill (No. 8) referred to in this paper is no longer made, it being an obsolete pattern, the present type being No. 18.

The point brought out in this paper that the force of the air feed on a stoper has a very material effect upon its drilling capacity is one not generally considered, but it is of the utmost importance in stoper service.

The Design, Construction, and Cost of Two Mine Bulkheads

BY SIDNEY L. WISE AND WALTER STRACHE,* NEW YORK, N. Y.

(Salt Lake Meeting, August, 1914)

WHILE the installation of mine bulkheads to retain water under high pressure is by no means a rarity, the following points which arose in the designing and placing of two of these bulkheads may be of interest:

The Hibernia magnetite mine, located about 40 miles west of New York, in the State of New Jersey, is partly filled with water. This mine is located on an ore lens, the outcrop of which is over a mile long. The ore has been mined from this lens from the surface to a depth of more than 1,500 ft. It was held desirable to separate the old workings from the new, and to allow the former to fill to the 850-ft. level. Two weak places existed between the new and old workings below this level: namely, a temporary bulkhead on the 10th level and a rock bulkhead of indeterminate thickness on the 16th level. When the matter of allowing the old workings to fill was investigated, the fact was developed that if these old workings were filled, the bulkhead on the 10th level would be subjected to a water pressure of about 50 lb. per square inch, and that 200 lb. per square inch would act on the 16th-level bulkhead. As the above-mentioned barriers were not deemed of sufficient strength to permit of these pressures, it became necessary to design and install new bulkheads.

Design

As the 16th-level bulkhead was required to withstand a pressure of 200 lb. per square inch, it presented some difficulties. A careful consideration of the various types of mine dams now in use led to the adoption of a design of the form of a truncated wedge. In this, the pressure side of the dam is of greater area than the back, so that the resultant action is similar to driving the wedge. By cutting generous skewbacks in the walls, roof, and floor, this type in reality becomes an invisible arch. The wedge feature tends to compress the materials in the bulkhead, thereby adding to its imperviousness.

* Non-member.

Concrete was chosen as the material. In order to lessen the labor and simplify the construction of the forms, straight forms were placed on both the front and back of the dam; the arch in this bulkhead is therefore invisible.

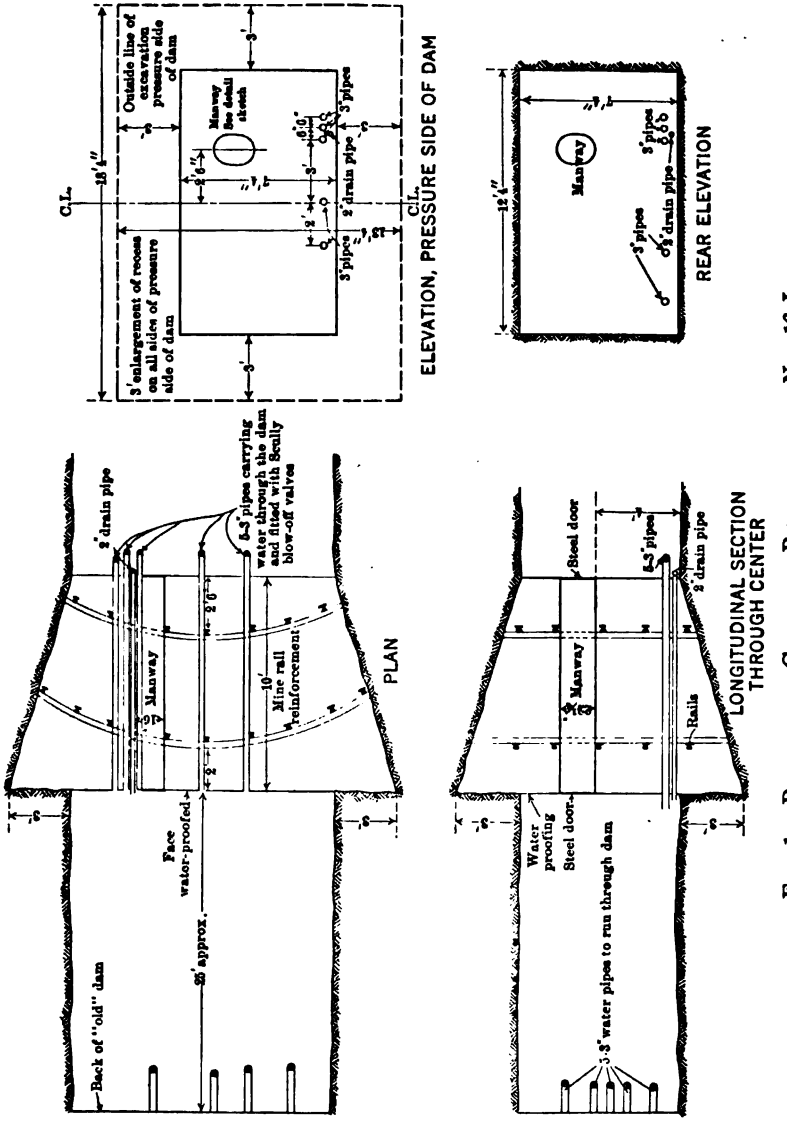


FIG 1.—REINFORCED CONCRETE BULKHEAD FOR No. 16 LEVEL.

As ordinary concrete is by no means impervious to water under the head of 200 lb. per square inch, it was decided to water-proof this bulkhead. The most economic method, consistent with good construction, was to face the entire pressure side with a 3-in. layer of a water-proofing com-

pound. As described in detail under "Construction," this facing was carried up with the concrete, thus insuring a perfect bond. The subsequent test proved the efficacy of this facing.

To provide for future contingencies, involving a possible further waterproofing of the pressure side, it was decided to place a manway through the dam, permitting inspection or repairs.

The relative positions of the bulkheads on the 16th level are shown in

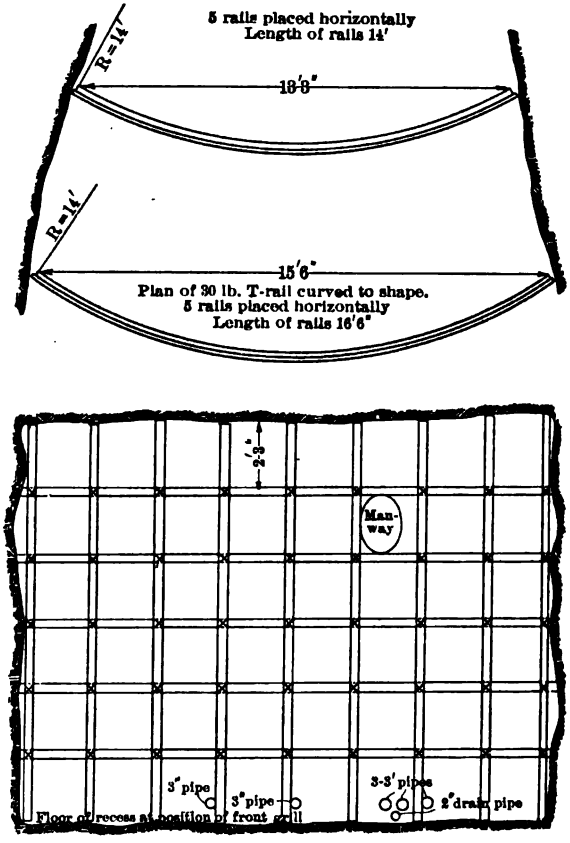


FIG. 2.—DETAIL OF REINFORCEMENT GRILL BUILT OF 30-LB. MINE RAILS FOR NO. 16 LEVEL BULKHEAD. RAILS SPACED AT 2-FT. CENTERS.

Fig. 1. Attention is called to the fact that the old bulkhead leaked to the extent of about 6 gal. per minute, and that five pipes pass through it for the purpose of draining the water in the old workings by pumping it to the surface from the 16th level of the new workings after it had passed through the old bulkhead. The new bulkhead was designed to continue this function of drainage in case it should be so desired, and for this reason prolongations of the pipes pass through it. Due to the

afore-mentioned leak, when the new bulkhead is completed and its manway sealed, the water which passes through crevices in the old bulkhead will soon fill the space between the old and the new dams. The new bulkhead will then be assuming the entire load.

The details of the steel reinforcement placed in the dam are shown in Fig. 2, the horizontal rails being curved to the radius of the invisible arch. Great care was taken to thoroughly coat all metal surfaces with mortar.

The design of the manway is shown in detail in Fig. 3.

It was decided to use a 1:2:4 mixture of concrete for this bulkhead. Atlas Portland cement was used, and a local sand, carrying less than 3 per cent. of foreign matter, was obtained. The broken stone employed

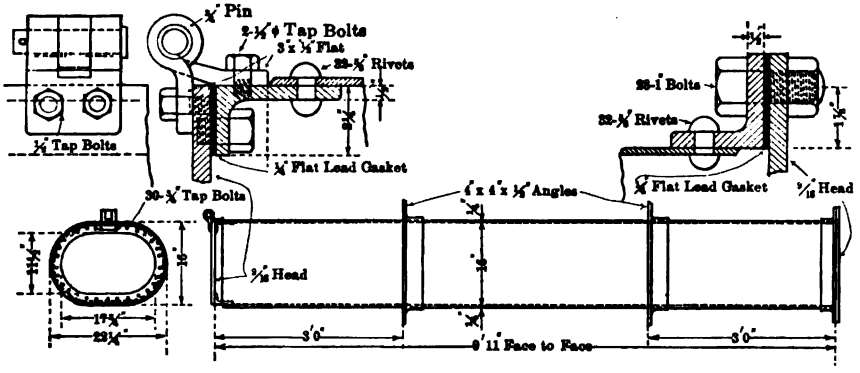


FIG. 3.—DETAILS OF MANWAY.

was a gneiss. This was the result of former milling operations, and was screened and washed before being used.

Construction

The construction of the 10th-level bulkhead presented no great difficulties. Ample storage space was available on this level and all the necessary materials were lowered and stored near the site. After thoroughly cleaning and washing the mushroom-shaped cavity, Fig. 4, the forms were placed, and braced from behind with 6-in. and 8-in. round timbers. The inside of the forms was covered with tar paper, and a 3-in. drain pipe, for possible future use, was run through the forms. This pipe was fitted with a gate valve on the working side. Two-thirds of a cubic yard comprised a batch of concrete, which was mixed rather wet, so that after a batch had been placed water rose slightly above the level mass. Three iron rails were placed across the mouth of the cavity for reinforcement. The 12 yd. of concrete were placed in 10½ hr.

In drilling the recesses for the 16th-level bulkhead, care was taken to

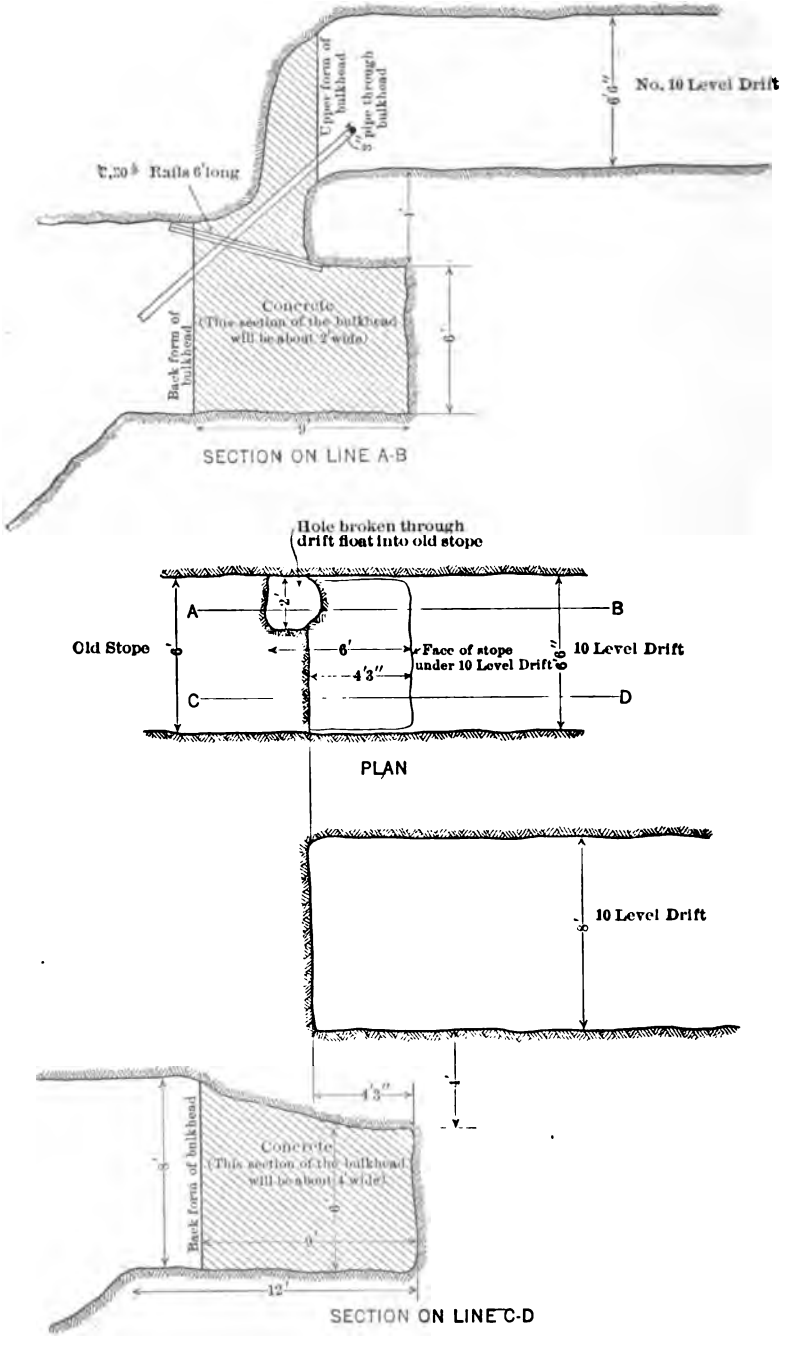


FIG. 4.—BULKHEAD AT NO. 10 LEVEL.

so point the holes that the excavation would coincide in form and dimensions to the design. At a distance of 25 ft. from the old dam, holes 36 to 39 in. in length and spaced 1 ft. apart were drilled in the sides, roof, and floor, at right angles to the course of the drift. Stoping and hand drills were used in this work and four men constituted the gang in this as well as the subsequent drilling with column drills. Thirty-five feet from the old bulkhead a series of holes 4 ft. in length and from 1 to 1½ ft. apart were placed slanting to conform approximately with the inclinations of the skewbacks. These holes were only burdened with about 1 ft. of ground. Under ordinary conditions longer holes would have been drilled, but the proximity of operating pumps made extraordinary precautions necessary for their protection during the shooting, and so heavy blasting was not attempted. A round of six holes was shot at a time. A third series of holes was drilled slanting to conform with the deeper portions of the recesses. When blasted, this series broke evenly at the line of the 3-ft. holes first drilled and the resultant recess conformed almost exactly to the figure determined upon, and the total excavation agreed with the original estimate of 60 yd. The muck was economically disposed of in a nearby chute.

The materials required for the work were unloaded and stored close to the mouth of the shaft. Due to the lack of space in which to store the materials on the 16th level, the matter of lowering and delivering the required materials without interrupting the work was one of the most troublesome obstacles encountered. Eight to twelve men were employed on the surface in sacking sand and stone while the excavation was in progress on the 16th level. The empty bags produced as the cement was used augmented the 200 old cement bags purchased for the sacking. While enough sand could be stored on the 16th level for this entire bulkhead, there remained insufficient room for the storage of the daily requirements of stone and cement. It was found advantageous, therefore, to employ a small night crew, who lowered much of the material required for the next day's work.

The ten curved rails were bent to a 14-ft. radius over a form in half a day. This was done on the surface.

The forms on the 16th level bulkhead were built of 2-in. undressed lumber and 6 to 10 in. round posts were used for studding and braces. The forms were thoroughly braced and were wired to stiffen them further. The interior faces of the forms were covered with tar paper, and the junction of the forms with the rock was plastered with a 1:1 cement mortar on all sides. The pressure-side forms were carried to the roof of the level at once, but did not extend into the recess.

The recess was thoroughly cleaned of loose rock and washed down, and all the reinforcing material, pipes, and the manway were placed in position before the concreting was started. Furthermore, the floor and sides

of the recess were plastered with a 1:1 cement mortar before placing the concrete.

The concrete was made of one part cement, two parts sand and four parts stone, these proportions being determined by actual measurement. A batch of concrete contained $\frac{2}{3}$ cu. yd. The sand was first placed on the mixing platform and the heaps flattened down. On this was emptied the cement, and these two materials were thoroughly mixed and flattened out before receiving the stone. This mixing took place about 12 ft. from the front form of the bulkhead. Enough water was used to make a wet mixture. Two men did the first mixing and turned the mass, then passed it on to the next two, who again turned it, passing the finished concrete to the last two men at the mixing board. These men shoveled directly into the form. In this manner, while each two men received a short rest of a few minutes between batches, fresh material was being placed on the starting end of the mixing platform while the men nearest the form were still disposing of the concrete mixture. This also insured a thorough mixing. One man remained in the form to level off each batch. The best day's work consisted in the placing of 12 yd. of concrete.

The water-proofing compound, "Impervite," was carried up as a 3-in. facing, its level being kept the same as that of the concrete. An even thickness of the water-proof layer was maintained by the use of three forms of $\frac{3}{16}$-in. plate, 6 ft. long by 6 in. wide, fitted at the upper corner with 3-in. spreading bolts. These forms, placed across the entire width of the face, were raised 3 to 4 in. at a time, and enough concrete was then shoveled against them to keep them in place. The almost semi-liquid water-proofing compound was mixed on the level and was carried to the forms in buckets.

Before leaving at night, sharp man-size (about 100-lb.) stones were set at least 6 in. apart in the concrete mass. This made a strong bond, and before concreting the next day this rough surface was freshly plastered with a thin 1:1 mortar.

As the roof was reached, false forms were placed, and the work was finally finished in tightly bonded dovetailed blocks.

Throughout the work, the leakage from the old dam, 6 gal. per minute, passed through the 2-in. drain pipe of the bulkhead.

Seven 2-in. grout pipes, four on the pressure side and three on the opposite side, were placed in the concrete as the work neared completion. They were all located near the roof and were directed to such places as were most difficult to fill with concrete. As the work had to be hurried, but a day and a half elapsed after completion of the cement work before grouting was begun. The grout mixture was a mortar consisting of one and one-half parts of sand to one part of cement made fluid with water-dissolved "Impervite." A mine-made grout "gun" was used, and the grout was forced successively into the several pipes by means of air

under the pressure of 85 lb. per square inch. As the grout was forced through the different pipes the ejection of some of this material through the other pipes indicated that the greater voids were filled. As the "gun" connections were changed those pipes giving the greatest discharge were plugged, and the discharge was finally limited to one pipe. This too, was filled and plugged. The first day's grouting was allowed to set over night, and the following day all the pipes were again tested. This time there was no communication between the pipes, and as little or no grout could be forced into any one of the pipes the grouting was considered most satisfactory.

Three weeks were determined upon as the period which should elapse before the new bulkhead should receive any load. During this time the 2-in. drain pipe was left open.

At the expiration of this time the completed 16th-level bulkhead was tested by pumping water up to the pressure of 160 lb. per square inch into the space between the old and new bulkheads through the 2-in. drain pipe. The results were entirely satisfactory, as the total seepage amounted to only ½ gal. per minute at first. This small leakage subsequently stopped almost completely.

Cost

A cheap class of labor was employed exclusively, the men receiving $2 per 10-hr. shift.

Following are tables showing the cost of the work. The interference caused by the necessity of keeping two large pumps in operation within 50 ft. of the 16th-level bulkhead was perhaps the greatest cause for the apparent high cost. The labor cost of lowering materials was also very high for the amounts handled, which, in the case of the 16th-level bulkhead, had to be lowered 1,350 ft. in one skip.

Summary of Costs

Division	16th-Level Dam	10th-Level Dam
Labor	$790.00	$134.00
Superintendence	130.00	30.00
Transportation	50.46	4.50
Materials	503.88	37.23
Totals	$1,474.34	$205.73

Costs per Cubic Yard

Division	16th-Level Dam	10th-Level Dam
Labor	$13.17	$11.15
Superintendence	2.17	2.50
Transportation	0.84	0.37
Materials	8.38	3.10
Totals	$24.56	$17.12

Methods and Economies in Mining

BY CARL A. ALLEN,* DENVER, COLO.

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*Non-member.

Introduction

In any discussion of mining one is repeatedly confronted with the difficulty of dealing with so many variable conditions. It is not an exact science and in the choice of a method each varying factor has a certain weight, which, in many cases, experience alone can determine. In mathematical terms, it is a function of many variables.

A discussion of mining also loses much of its value unless costs are considered, because the expectation of profit is the only excuse for carrying on mining at all. As conditions vary they cause fluctuations in cost and there are few operations to which a definite cost can be assigned. The character of the ore may make it difficult to drill, yet because of the ease with which it breaks the total cost of drilling and blasting may be low.

In preparing this article the attempt has been not to cover the whole field of mining, but to describe the different methods of stoping and mining which have basic principles. In addition to this an effort has been made to show the advantage of dissecting the methods into their detailed operations and applying to these a mathematical study as an aid to the judgment in determining which is the best method to adopt, or in attempting to reduce the cost of a method already in use.

Methods of mining include stoping, caving, and various methods of working large deposits which, in addition to the method of actually breaking the ore, require elaborate and definite plans of development of the orebody. The ordinary methods of stoping are too familiar to all for any elaborate discussion, but it has seemed advisable to review the subject and give the principal advantages and disadvantages of the different methods.

Factors Affecting Choice of Method

In addition to the various external factors, such as the supply of labor and the financial status of the operating company, the principal items that influence the method of mining to be adopted are:

The size and shape of the deposit.
Character of the ore, whether high or low in grade.
Whether the values are regularly or irregularly distributed.
Physical character, whether hard or soft, tough or brittle.
Character of the country rock.
Immediate and future demands for ore.
Amount of development work done or that may be necessary.
Amount of water to be handled.
Cost of power, timber, and supplies.
Ventilation.
Whether drilling is to be done by hand or with machine drills.

Review of Stoping Methods

Depending on these factors, the following methods of stoping may be employed:

Underhand stoping:
- Ore is hoisted to the level above; Cornish stoping.
- Ore is drawn from the level below.

Overhand, or back stoping:
- Starting stopes.
- Drift stoping.
- Cutting out, or lead stoping.
- Raise stoping.
- Longitudinal back, flat-back, or long-wall stoping.
- Rill-cut stoping.
- Saw-tooth back stoping.
- Shrinkage stoping.

Combination stoping.
Breast stoping.
Side stoping.
Sublevel stoping.
Square-set stoping.
Filling methods.

Underhand Stoping

The method of underhand stoping in which the ore is drawn to the level above (*B*, Fig. 1), is called Cornish stoping. It finds application only when it is necessary to mine a lens of good ore below a level and it is not practicable or financially possible to do the necessary development to come up from underneath. Its disadvantages are the excessive cost of raising both ore and water.

Underhand stoping where the ore is drawn from the bottom (*A*, Fig. 1), has more merit than is usually accorded it, especially in the Western States. On the Rand it has been used almost exclusively. Its advantages and disadvantages as compared chiefly to overhand stoping are as follows:

Advantages:
- Ease in drilling down holes when drilling is done by hand.
- Holes are drilled wet and dust is eliminated.
- No platforms are required on which to drill.

Disadvantages:
- Limited to steeper pitches than overhand stoping because the ore does not work straight down the foot wall to the raise; in flat pitches this would necessitate more shoveling.

Limited to good walls if the vein is steep. Loose rocks slough off from poor walls and endanger the workmen below.

Levels ordinarily must be driven closer together to reduce the height of the unsupported walls; this necessitates an added cost of development.

Raises must be put up at frequent intervals. This work in some cases amounts to 35 or 40 per cent. of the total development.

No ore reserves are possible.

The waste that can be sorted in the stope if the vein is steep is limited to what can be thrown on lagging supported by stulls.

Certain efficient types of stoping drills cannot be used.

The great advantage is, of course, the drilling of down holes when drilling is done by hand. Few men to-day can or will drill very many

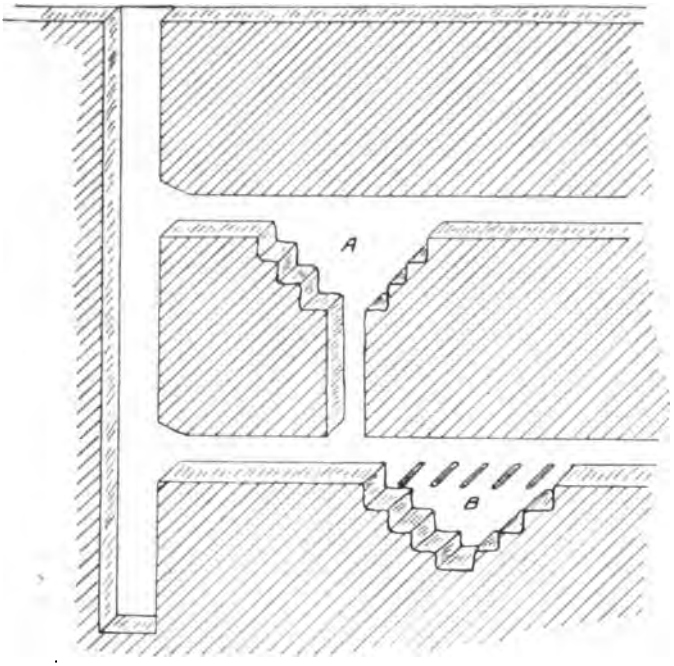


FIG. 1.—UNDERHAND STOPING.

"uppers" in a shift. South Africa had its native labor which could not be taught to drill them. It might seem on first thought that the disadvantages are so numerous that they preclude any chance of the method being adopted under ordinary labor conditions, but I have examined mines where the combination of conditions in narrow veins indicates very strongly that the method would be more economical than overhand stoping and quite as safe.

Overhand Stoping

Overhand stoping, in general, has the following advantages and disadvantages:

Advantages:

Stoping can be started directly from the level without any raises or winzes.

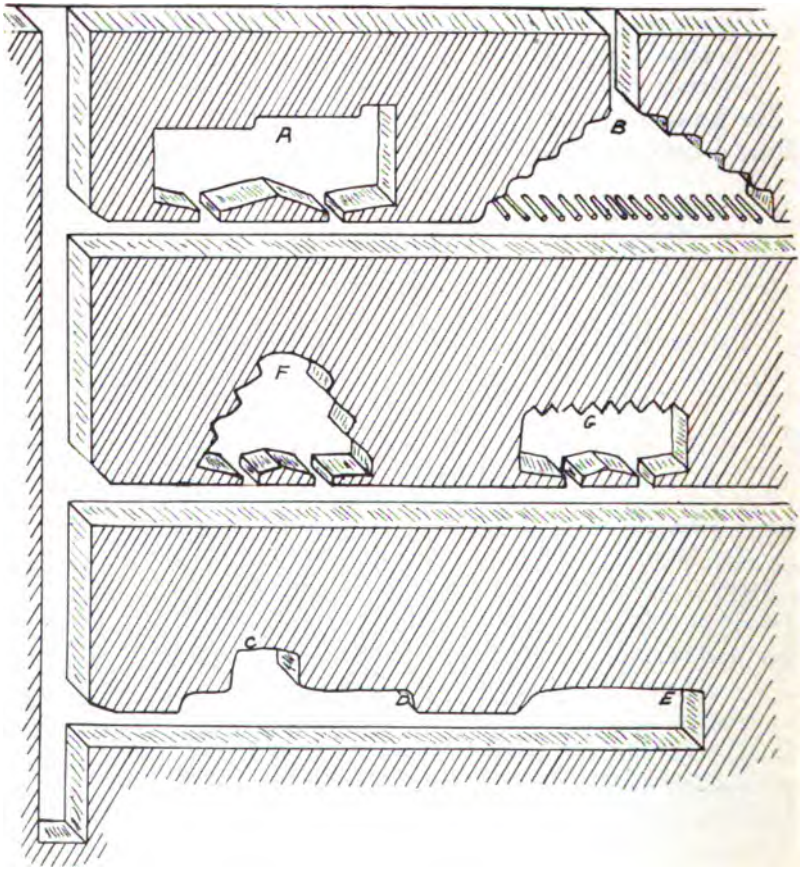


FIG. 2.—OVERHAND STOPING.

Levels can be driven far apart. These two considerations mean less cost for development.

Advantage is taken of the force of gravity in breaking the rock.

Miners work at the back and can inspect it so that the danger of falling rocks is largely eliminated.

Either ore or waste can be stored in the stope.

A flatter seam can be worked than in underhand stoping without

the necessity of shoveling, as the ore runs directly down the pitch and is given an impetus by the blast.

Water takes care of itself.

Pillars of ore or waste can be left easily.

Disadvantages:

In most cases holes must be drilled up or flat.

If the stope is not filled with ore or waste, stulls and platforms must be used, even though the walls require no support.

If the raises are far apart ventilation is poor.

Starting an Overhand Stope.—In Fig. 2 are shown various methods of working the back in overhand stoping. To start the stope, if the ore is low grade and timber is scarce, a pillar or pillars are left above the level as shown in *A*, and if the ore is high grade it is all removed above the level and stulls are placed as shown at *B*. To start a stope as shown at *B*, a cutting out or lead stope is broken immediately above the level as shown at *D*, Fig. 2. *C*, Fig. 2, is a wide raise or raise stope, which is one method of opening an overhand stope, and *E* is a drift stope, which is a term used in the Lake Superior copper mines and means a wide and high drift as a start for the overhand stoping.

Longitudinal Back, Flat-Back, or Long-Wall Stoping

After starting an overhand stope the shape in which the back is carried is often of prime importance. In *A*, Fig. 2, is illustrated the method of carrying a flat or longitudinal back (the term long-wall is used sometimes when the vein lies nearly flat). In this method the benches or breasts are frequently made of a height sufficient to allow of a square set being placed. This is done at Butte and it is often referred to as breast stoping. In general, a flat-back stope as illustrated has the following advantages and disadvantages:

Advantages:

If the stope is filled with waste or ore it is more convenient for men to work on a level surface. Tramming, shoveling, or sorting in the stope can be done to better advantage, and waste raises can be driven farther apart.

Square sets may be conveniently placed by making the benches the proper height.

Timber struts or cribs can be used between the filling and the back, in case the latter is weak.

A long stope distributes the broken ore well along the level, and tramming is thus facilitated.

Disadvantages:

The principal disadvantage of a flat-back stope is that if filling is used, and kept close to the back, it must be distributed in wheelbarrows or cars.

Rill-Cut Stoping

Rill cut, or rill stoping, as shown at *B*, Fig. 2, has the principal advantage that when filling is used it can be run down through a raise at the apex of the stope and will fill the stope without shoveling. Its disadvantages, on the other hand, are as follows:

When waste filling is run in from the raise it has a sloping surface which makes it difficult to keep ore and waste from mixing, and is unhandy for men to work.

Raises must be put in at frequent intervals, and the added cost of these raises may exceed the cost of spreading the filling in a flat-back stope.

Ore chutes must be carried up through the filling, and the timber used in their construction cannot be recovered. This is a disadvantage as compared with shrinkage stoping.

At *F*, Fig. 2, is shown a method of rill stoping in which the benches are inclined so that down holes can be used. Down or water holes give off no diagreeable dust, and can be drilled faster with piston machines.

Saw-Tooth Back Stoping

This method of carrying the back in an overhand stope is shown at *G*, Fig. 2. It is claimed that it makes drilling more convenient if machine drills mounted on a bar are used.

Shrinkage Stoping

Shrinkage stoping refers to any overhand method in which the stope is kept full of broken ore until it is completed. The miners stand on top of the broken ore and work at the back. As broken ore takes up more space than ore in a solid mass, about 35 to 40 per cent. of it must be drawn to leave room for working.

Advantages:

Raises may be placed far apart.

The broken ore serves as a support to the walls. This does away with the necessity for much timber.

The miners work on top of the broken ore; timber platforms are eliminated, and the work is made much easier. It is also convenient to work a larger number of men in the stope.

A large ore reserve is maintained.

Large rocks can be broken with sledges in the stope and blocking the chutes is avoided.

No ore passages are required from the level up to the back of the stope. Manways are necessary up through the broken ore, but the

timber used in their construction can usually be recovered when the ore is drawn.

Ventilation is better than in an empty stope.

Disadvantages:

Stoping must be kept ahead of the demand for ore. This requires additional capital.

There is practically no opportunity to sort ore in the stope.

Ore filling is not permanent and the stope may cave when it is withdrawn.

Scaling or unstable walls may cause waste to mix with the ore or prohibit the use of the method altogether.

Although shrinkage stoping is rather generally used, it would be used more if it were not for the fact that many mines are not in a condition to keep an ore reserve, but must draw the ore for the mill as fast as it is broken. The added efficiency to be obtained from the miners when working on a firm floor of ore, and not on loose lagging laid on stulls, is a very important advantage of the method.

Combination Stoping

In a discussion of stoping methods, steeply pitching veins are usually referred to, because the majority of veins in nature are steeper than 45°, and also because to refer each method of stoping to veins of all dips causes confusion. Before considering combination stoping, however, a brief *résumé* of the methods of handling ore in veins of different dips is necessary. In veins dipping from 35° or 40° up to 90° the ore runs down by gravity. From 20° to 35° it must be assisted by shoveling, or by using chutes with smooth bottoms or placed at an angle steeper than that of the vein. From 10° to 25° the ore may be taken down to the level by shoveling, shaking chutes, mono-rail trams, gig-back railways, or conveyors. From veins that are horizontal up to those having dips of 10° to 15° tracks are usually laid on the foot wall and cars are pulled up to the face by men or animals. An animal can pull an empty car up a 6° slope and a loaded car up a 3° slope. So in dips greater than 6° the track must be laid at an angle with the line of the dip.

In combination stoping, which is illustrated in Fig. 3, and which is a combination of underhand and overhand stoping, it is possible to keep the working face more nearly in a line parallel to the raises and perpendicular to the level. This is a distinct advantage if a stationary or shaking chute is being used to carry the ore to the level, because a large part of the face of the stope is accessible to the chute. This becomes a double advantage if a large output is required. In combination stoping development work is also reduced because levels may be driven far apart, and fewer raises are required than in simple underhand stoping.

Side Stoping

This is an extreme of combination stoping in which the working face of the stope is vertical. The face is parallel to the raises just as the face in flat-back stoping is parallel to the levels.

Breast Stoping

Breast stoping refers to the working of a flat orebody, or a flat section of an orebody, just as coal is mined from a flat coal seam. That is, a slice is worked in a horizontal direction. The assumption is that there is no open stope either above or below the slice, or else the method

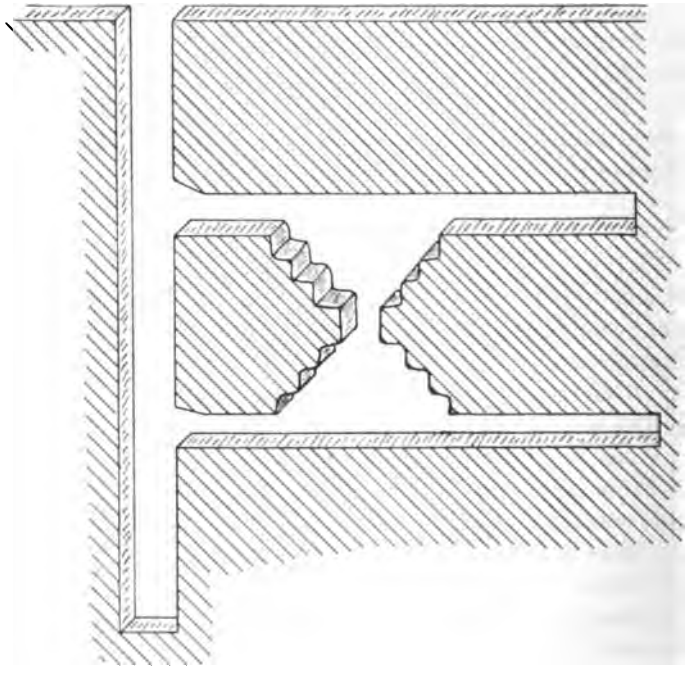


FIG. 3.—COMBINATION STOPING.

becomes either underhand or overhand stoping. In some cases the benches in underhand or overhand stoping are called breasts, and the method breast stoping, but I believe that is not the usually accepted definition. Breast stoping, in the strict sense, is the only name applicable to the mining of a horizontal slice, such as the sill floor of a large overhand stope, or the slices in the top-slicing method.

Sublevel Stoping

This method, described by F. W. Sperr in the *Engineering and Mining Journal* of June 5, 1912, and by P. B. McDonald in the *Mining and*

Scientific Press of July 5, 1913, is really a combination of several different methods of stoping. Fig. 4 is an illustration of the method, and shows half of the vein cut away where the stoping is being done. The miners go from the haulage level up the raise into the sublevels. On the first sublevel they will blast into the shape of funnels the raises that come up from the haulage level, and after working back the slice, *s*, by breast stoping will drill holes at *a* and shoot part of this bench down into the raises. On the second sublevel the miners work back the slice, *s*, and then with down holes at *b* will blast the remainder of the first bench down into the raises, and with uppers at *c* shoot off part of the second bench. Both sets of holes will be fired at the same time. In this way

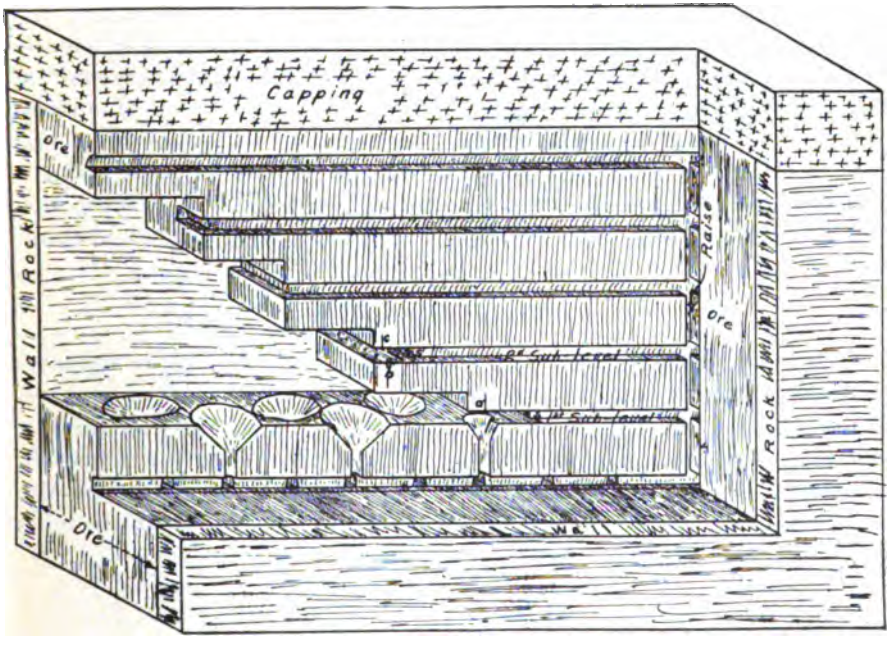


FIG. 4.—SUBLEVEL STOPING.

each sublevel is drawn back; the broken ore is drawn off through the funnel-shaped raises, which leaves a large open stope. The sublevels are driven about 25 ft. apart vertically, and 8 to 10 ft. are shot off from each bench from the sublevel below. Obviously where there is a capping above the ore, as shown in the cut, part of it will fall when the upper bench is blasted and part or all of this ore may be lost. "When the ore is from 50 to 100 ft. wide, a narrow stope of about one-third of the width of the ore is drawn back through the middle, leaving a pillar of ore standing on each side." These pillars or slabs left against the walls are then drawn back in the same manner. The method is applicable to

veins from 12 to 100 ft. wide, or wider, if pillars are left between stopes, and evidently comes into competition with shrinkage stoping, square-set stoping, and top slicing. It requires expensive development but permits a large tonnage to be broken in a relatively small stope. The working back of the slices on the sublevel is expensive mining, but after that is done the rest of the ore blasted from the benches requires a comparatively small amount of powder.

Square-Set Stoping

Square-set stoping may refer to any method of stoping in which square-set timbers are used. Unless otherwise specified, however, the term is limited to overhand flat-back stopes in which square sets are used, either with or without waste filling. Square-set mining is advantageous when the vein is wide and the walls will not stand without timbering and shrinkage stoping cannot be used; when, on account of surface conditions, caving cannot be allowed, or where caving might cause the loss of other orebodies; when the ore changes rapidly in grade and requires frequent sampling; when the orebody is irregular in outline; and when old stopes may have to be approached or passed through at some later time.

Filling Methods

Mine workings are filled with waste as an aid to timbers in supporting weak walls or back, or to avoid fully or in part the use of timbers. Workings that must be prevented from caving for a length of time in the future are best protected by waste filling. The filling may be waste rock from development, rock blasted from the walls or surface for the express purpose of filling, or, if available, sands from concentration mills make excellent filling and are cheaply placed in the stopes.

Caving Methods of Mining

There are three distinctive methods of working large deposits that involve, in some way, the factor of caving. Slightly different names are in use, but those that are simple as well as descriptive are top slicing, sublevel caving, and block caving. In a discussion of these methods, three things must be borne in mind: that the methods are usually applied to massive deposits; that the deposits are usually divided into blocks or panels, and the description of mining one panel is practically a complete description of the method; and that, in most cases, the orebodies do not come to the surface but are covered with a capping. This capping may be glacial drift, as in some instances in the iron regions around Lake Superior, or it may be rock from which the ore values have been leached, as is the case at some of the large copper deposits.

Top Slicing

Top slicing, illustrated in its ideal form in Fig. 5, consists in the working of an orebody in horizontal slices, beginning at the top. Levels, for haulage, are established at proper intervals and, from these, raises are put up to the top of the orebody about every 50 ft. Starting at the tops of these raises drifts are run out and then, retreating toward the raise, the slice is worked back by breast stoping. The overburden or

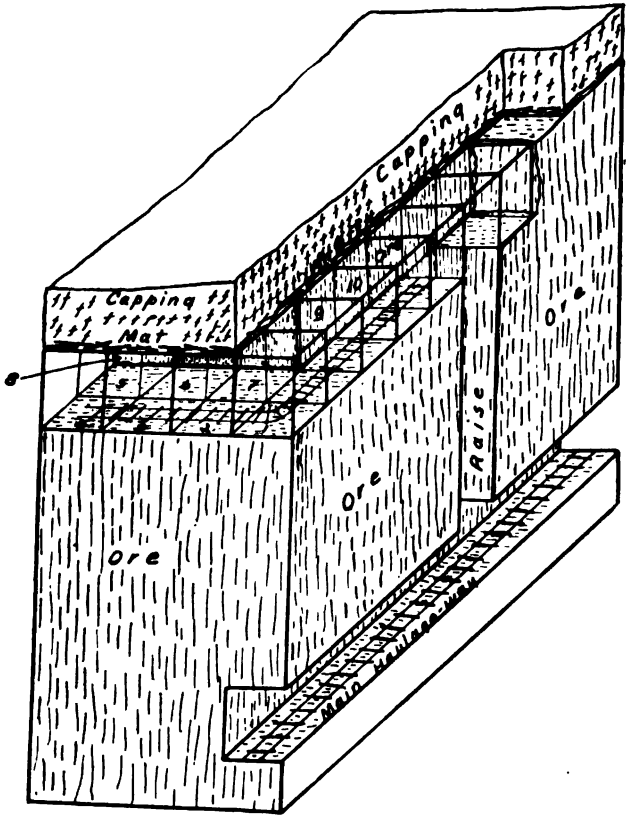


FIG. 5.—TOP SLICING.

capping lying above the ore is supported on square sets or posts until a slice, or part of a slice, has been worked. The floor of the slice is then covered with slabs or plank and the supporting timbers are shot out and the capping is allowed to cave on to the timber floor. This floor and the broken posts form what is called a mat, which keeps ore and capping separated. The miners then start in the raise and work out another slice of the ore just under the previous one and catch up the timber mat with posts or square sets. When this slice is completed another

floor is laid and the supports again blasted. In this manner the orebody is worked in successive slices from the top downward. The broken ore from each slice is run to the raise in wheelbarrows or cars and dropped to the haulage level. The capping caves and follows down on top of the mat.

In this method the ore is not caved at all, but the ground above the ore does cave, and the surrounding country will cave more or less according to the amount of ore removed. If there are any other orebodies in the region affected by the cave, they will be lost or their recovery made more difficult. This is one of the reasons which may prohibit the use of caving methods. Also, unless the deposit is small compared to the distance to the surface, surface subsidence will take place and the surface must be free from buildings or roads. Top slicing is adaptable to heavy ores that can be mined easily, and require heavy timbering and filling if worked by overhand stoping. Deposits of large extent, over which the overburden will cave readily, are customary conditions. In some cases, top slicing may require as much or even more timber than an overhand stope with square sets, but for very heavy ore, which if worked overhand necessitates strong sets, reinforced and braced, it takes less timber, and in any case poorer timber may be used.

The following are the advantages and disadvantages of top slicing, chiefly as compared with overhand square-set work or with other methods of caving.

Advantages:

Cheaper timber, and for heavy ores less timber, is required than for square-set stoping.

No waste filling is necessary.

Ore can be sorted as mined.

Very little waste from the capping becomes mixed with the ore.

Less skilled labor is required.

Any rich fine ore produced in breaking will be saved on the slice below.

Complete extraction of the ore is possible.

Disadvantages:

Caving may cause injury to surface structures or render unworkable other orebodies in the region.

Mining the ore by breast stoping requires more drilling and blasting.

Broken ore on the slice must be shoveled into barrows or cars and taken to the raises.

Development must be kept ahead of the demand for ore.

It is difficult to leave bodies of low-grade ore which may be found in the deposit.

Ventilation is difficult, and in some cases, the decaying timbers in the mat give off heat and obnoxious gases.

It is possible to work only on the top of the ore, and although several slices may be worked at the same time, it is not possible to establish work at different levels. This may make it difficult to obtain a required tonnage from a deposit of small horizontal section. Capping should cave readily, and not arch, or the method may become very dangerous.

Sublevel Caving

This method, otherwise referred to as sublevel drifts and back caving, or sublevel slicing, resembles top slicing in that the orebody is worked from the top down, and the ore is taken out in horizontal slices. A block

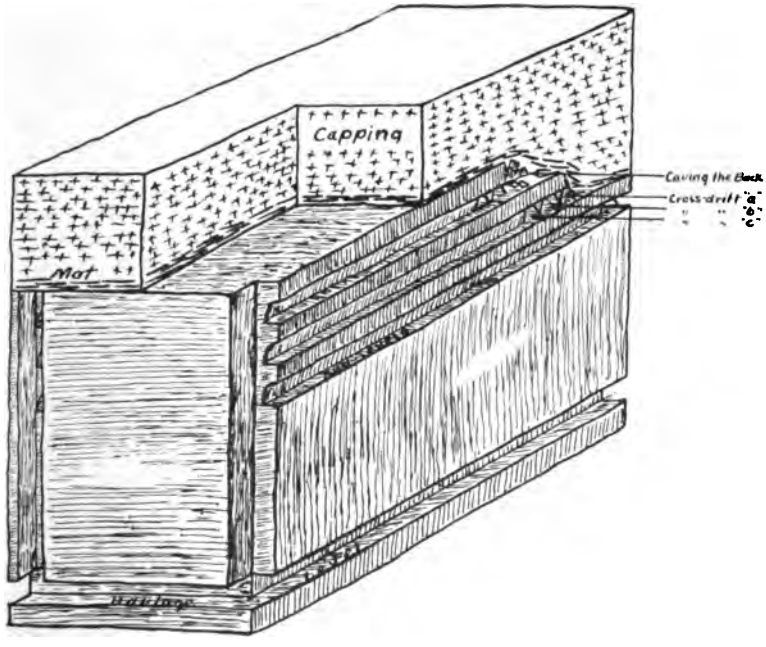


FIG. 6.—SUBLEVEL CAVING.

of ore is left above the slice, however, and when a small portion of the slice has been mined out, this back is allowed to cave. Fig. 6 shows a block or panel of ore worked by sublevel caving. Raises are put up from the haulage level, and from these sublevel drifts are driven 14 to 20 ft. apart vertically. When these drifts reach the boundary of the property, or of the panel to be worked, cross drifts as *a*, *b*, and *c* are driven across the block and timbered. The back of ore is thus undercut but is supported by the drift timbers. The timbers of cross drift *a* are next blasted and the weight of the capping caves the back of ore down on to the floor

of the slice, where the miners shovel it into cars and push it to the raise. A timber mat may be used as in top slicing. Sublevel caving is adaptable to massive deposits or wide veins in which the ore is not difficult to break, yet is firm enough to hold the capping while supported by the drift timbers. An overburden that will readily cave is necessary.

Advantages:

The cost of mining is low, because a large percentage of the ore is broken with little or no blasting, and the amount of timber used is small.

A large output is possible.

A large percentage of the ore can be saved.

The ore can be kept freer of waste than in block caving.

Disadvantages:

It is limited in application to certain ores.

A large amount of development is required which must be kept ahead of the demand for ore.

A large part of the ore must be shoveled.

Caving endangers surface and other ore deposits.

There is some danger to the miners.

Ventilation is difficult.

Block Caving

Block caving is an extreme case of sublevel caving in which instead of a back of ore 5 to 10 ft. thick, one 50 ft. thick is undermined and allowed to cave. The method is illustrated in Fig. 7. After the bottom of the block, or panel, is cut up into pillars by drifts and cross drifts, the pillars are robbed, and then the remaining stumps are blasted out with one large blast. This allows the entire block above to cave. In settling it disintegrates so that it can be shoveled with very little additional blasting. For a block to cave it is usually necessary for it to be freed on one or more sides. This is done, as shown in the cut, by narrow stopes, called isolating stopes. After the ore has settled for from two to six months timbered drifts are driven through it. The broken ore is allowed to run into these drifts, starting farthest from the shaft, is shoveled into cars and trammed out. As soon as capping shows at any point shoveling is stopped.

The advantages of this method are:

The cost of mining is low because very little drilling and blasting or timbering is required.

The amount of development is small.

A large output is possible.

Disadvantages:

The method is limited in application to low-grade ore of such character that it will cave and disintegrate.

A large amount of ore becomes mixed with capping and lost. There is no opportunity to select or sort the ore.

Back Caving into Chutes

There is another principle that should be mentioned under mining methods. It is called back caving into chutes or chute caving and in some ways resembles block caving. (The description of a mine employing this principle is given in *Mining Without Timber*, by Brinsmade, p. 181.) Large overhand stopes are worked, not by drilling and blasting the entire back, but by blasting out narrow isolating stopes around the edges of the

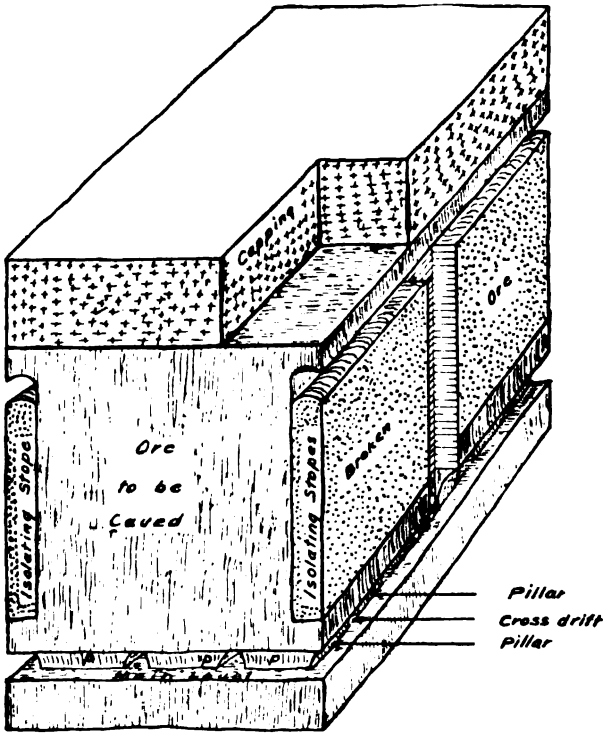


FIG. 7.—BLOCK CAVING.

main stopes and allowing the rest of the ore in the center to fall of its own weight. That is, a large percentage of the ore is mined by caving. The broken ore is drawn off through chute raises in the bottom of the stope.

As far as I am aware, the preceding pages cover the methods of underground mining that have distinctive or basic features and can be readily classified. There are innumerable different systems of mining but they all involve only these principles, or modifications and combinations of them.

Costs of Mining Methods

To compare properly two different methods of mining, it is not wise to attempt to calculate the total cost of each, but rather to compare the

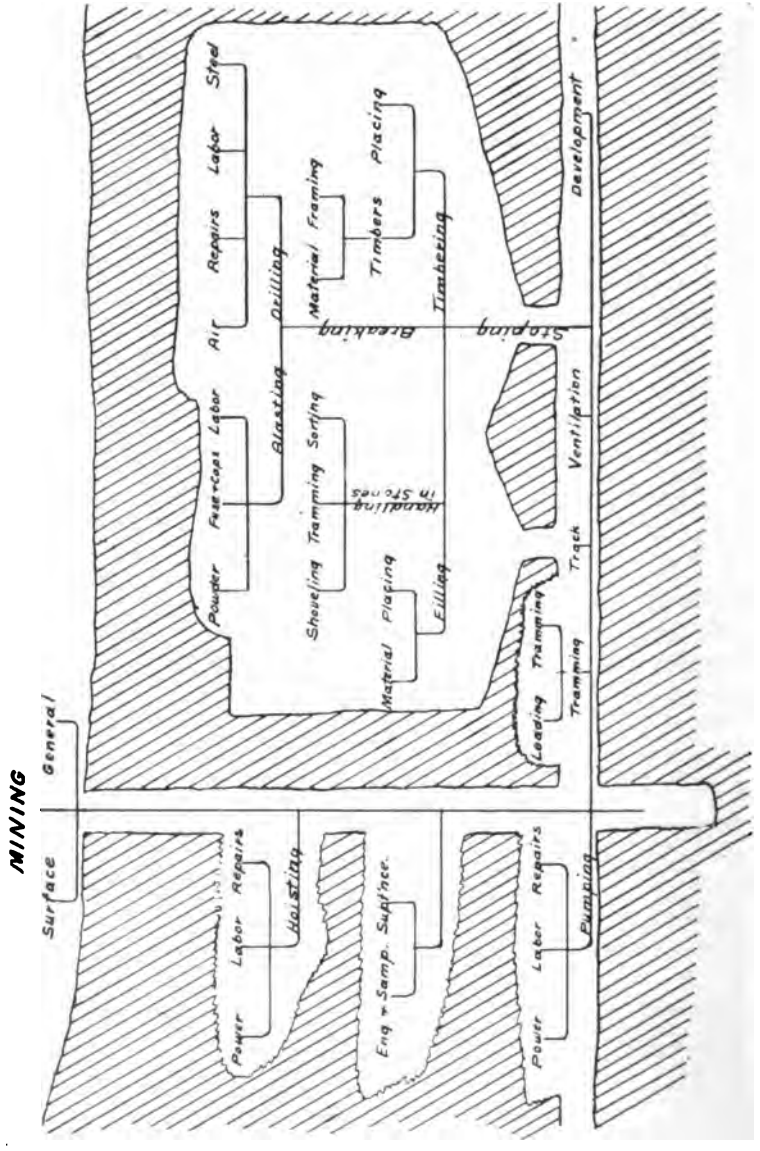


Fig. 8.—Diagram showing Arrangement of Mining Operations.

costs of those operations that are differently executed in the two methods under consideration. When any method is in operation the proper way to reduce the total cost is to dissect it into its different operations and see

if the cost of one or all of these cannot be reduced. Keen competition has forced large manufacturing and other industrial enterprises to give careful attention to cost accounting and efficiency engineering. This calls for the investigation of the smallest details; conditions are adjusted so that a workman will not lose time in going after a tool or in walking from one machine to another. An increase in the efficiency of a number of small operations means a corresponding decrease in the total cost of production. The reasons why efficiency engineering has been so generally adopted are two: first, it attacks its problem in a scientific manner and, second, it brings results in dollars and cents. Mine managers are not excusable from applying its principles just because the ore may be high enough in grade to pay dividends even under lax conditions.

The different operations of mining can be separated in different ways. Fig. 8 is a diagram which shows a logical arrangement and the smallest operations shown in it could be subdivided into yet smaller details *ad infinitum*. In determining upon a new method or in attempting to reduce the cost of one already established, these operations must be carefully studied. If modifying a method will increase the efficiency of one operation without affecting the rest, the problem is a simple one, but usually more than one operation is affected, some adversely, and some to advantage, in which case to determine which is the cheaper method it is necessary to apply to each operation its cost under the different conditions.

Costs of Drilling and Blasting. Comparison of Stoping Methods

As an appendix to this article I have collected a number of costs of mining as published in the reports of mining companies.

Obviously but few of these reports show the costs of the operations given in Fig. 8, and in order to bring out their detailed costs it is necessary to make certain assumptions. To bring out the costs of the different operations of drilling and blasting, consider the data on North Star, Alaska-Treadwell, and the Rand as given in the appendix, with general data on Cripple Creek and the Michigan copper mines. All are straight overhand stoping methods with the exception of the Rand, and, although not so stated, I assume that these data refer to underhand stoping. The first consideration is drilling. Up to the present time for underhand work, mounted machines were the only ones available, but it is probable that in the future they will be largely replaced by light unmounted machines of the Jackhamer type. For overhand work, the hammer stoping drill, unmounted, has demonstrated its superiority in reduced labor cost, increased ease of handling, and greater drilling speed in rock that is not too hard. While actually at work machines will drill from 0.25 to 15 in. a minute, usually 1 to 3 in. At the North Star mine, evidently, 25 to 30 ft. are drilled for each machine shift. Light stoping drills are used and break 7½ tons in a 4-ft. stope. In Cripple Creek the average

of several mines is 60 ft. for each machine shift. Light stoping drills are used and the machine runner is using the machine about 6 out of every 8 hr. In a 5-ft. stope 14 4½-ft. holes will break about 15 tons. In the Lake Superior copper region, in the amygdaloid lodes, machines drill from 30 to 47 ft. a shift in different mines.[1] In the conglomerate, on account of more massive copper and higher quartz content, about 24 ft. for each machine shift is the average (9-hr. shift) and seldom is more than 28 ft. drilled. Holes in both cases are usually 8 ft. deep and a burden of 2 to 3 ft. is placed on them. In the conglomerate, although harder to drill, 54 tons are broken for each machine shift, whereas in the amygdaloid only about 38 tons are broken. The stopes are 10 to 30 ft. wide. At the Alaska-Treadwell each machine in the stopes drills 28 to 34 ft. and breaks 32½ to 45 tons. Heavy piston drills are used and the stopes are about 60 ft. wide. On the Rand large machines break 7.2 tons in a 55-in. stope, 10 tons in a 6-ft. stope, 19½ tons in a 7-ft. stope, and small machines break 3.7 tons in a 4-ft. stope. To arrive at the actual cost of drilling it is also necessary to consider the cost of compressed air, machine labor, repairs, and drill steel and sharpening. The cost of compressed air varies from $0.40 to $2.50 for each machine shift. It depends on the cost of power, size and efficiency of the compressor, size and efficiency of machine drill, and the conditions of the drill. The accompanying table taken from *Ore*

			Cost per 1,000 Cu. Ft. Free Air Compressed			Cost per Drill Shift.		
Style of Compressor	Maximum Capacity Cu. Ft. Free Air per Minute	Total Cost per H. P. Hour	Sea Level	5,000 Ft. Altitude	10,000 Ft. Altitude	Sea Level	5,000 Ft. Altitude	10,000 Ft. Altitude
Simple steam (non-condensing.....	200	2.2c.	5.9c.	5.3c.	4.8c.	$2.07	$2.22	$2.40
Compound steam non-condensing).....	300	1.5	4.0	3.6	3.3	1.40	1.50	1.65
Simple steam (condensing).....	2,500	1.0	2.7	2.4	2.2	0.95	1.01	1.10
Compound steam (condensing).....	3,000	0.8	2.2	1.9	1.8	0.76	0.81	0.88

Table based on 8-hr. shift with coal at $5 a ton.

[1] Claude L. Rice: *Engineering and Mining Journal*, vol. xciv, No. 5, p. 217 (Aug. 3, 1912).

Mining Methods, by W. R. Crane, gives the cost of operating a 3-in. drill in granite.

At the North Star mines machine men cost $3 a shift or 43c. a ton; machine power, 40c. a shift or 6c. a ton; repairs and lubrication, 42c. a shift or 6c. a ton; drill steel and sharpening, 75c. a shift or 10c. a ton; total for drilling, $4.57 a shift or 65c. a ton. To this must be added 25c. a ton for powder, 4c. for fuse and 1c. for caps, which makes a grand total of 95c. a ton. Allowing for rock left in the stopes this would be reduced about 15 per cent.

At Cripple Creek machine men cost $3.50 a shift or 24c. a ton; air, $2 a shift or 13c. a ton; repairs and lubrication, 25c. a shift or 2c. a ton; drill steel and sharpening, 50c. a shift or 3½c. a ton. Total for drilling, $6.25 a shift or 42½c. a ton. To this add 24c. a ton for powder, 3½c. for fuse, and 1c. for caps, which gives a grand total of 71c.

At the Alaska-Treadwell machine men and helpers average $6.85 a shift or 21c. a ton; air, 75c. a shift or 2c. a ton; repairs, 50c. a shift or 1½c. a ton; drill steel and sharpening, 45c. a shift or 1½c. a ton. Total for drilling, $8.55 or 26c. a ton. Adding 15c. for powder, 1c. for fuse, and ½c. for caps gives 42½c., to which must be added an additional 6c. for extra labor which includes powder men. This gives a grand total of 48½c.

The data given in the appendix on operations on the Rand are too incomplete to analyze as the others are analyzed. From other data which I have, I assume that the costs are about as follows: Machine labor, $4.50 a shift; air, 75c.; repairs, 50c.; drill steel and sharpening, 50c. This is a total of $6.25. An average of 12 tons was broken each shift, which gives a cost of 52c. a ton for drilling. To this must be added 25c. a ton for explosives, which gives a grand total of 77c.

On the Rand, despite the greater stoping width, the cost of explosives is nearly as much as at North Star or Cripple Creek and the drilling cost per shift is much greater. The influence of the width of the stope on the tonnage broken is well illustrated in the Alaska-Treadwell and the added cost of machine labor is minimized. The long holes used and the character of the ore, no doubt, preclude the possibility of using one-man machines.

From the above, it is seen that an item worthy of consideration is that of drill steel and sharpening. It varies greatly but in any case is a factor worthy of attention. Not only the cost of new steel and the wages of the blacksmith, but the delivery of sharp drills to the stope and the removal of dull steel, should be considered. By comparing cost items occurring in the reports, which are given in the appendix, other features influencing drilling and blasting might be shown, but the greatest value results in making comparison when conditions are similar and are known with exactness.

Lighting

It is stated in one of the mining books that the cost of lighting for each ton of ore mined is about the same in all mines. This is not the case. The cost varies according to the light used and the number of men underground. The entire subject of mine lighting has been well covered by Frederick H. Morley in the *Mining and Scientific Press* of April 11, 1914. The following table taken from this article gives the cost of acetylene lighting in 10 of the large metal mines.

Name of Company	Number of Men Employed Underground	Number Using Lamps	Carbide Consumption per Lamp, Oz. Shift	Cost Carbide per Lb., Cents	Cost per Lamp Shift, Cents	Cost of Candles per Shift, Cents
Homestake Mining Co.....	1,025	1,025	8.0	3.50	1.75	7.00
Ray Con. Copper Co.....	1,400	1,200	9.0	4.50*	2.50*	5.00*
Quincy Mining Co.....	1,389	575	6.7	3.50*	1.46*	.....
Osceola Con. M. Co.	625	625	6.0	3.50	1.38	.....
United Verde Copper Co....	600	575	6.5	5.50	2.23	5.40
Bunker Hill & Sullivan Co..	460	208	7.0	5.25	2.30	6.18
Calumet & Arizona M. Co...	1,000	60	7.0	5.50	2.40	6.64
Ohio Copper M. Co.....	96	37	8.0	5.80	2.90	.....
Nevada Con. Copper Co....	200	20	4.0	4.67	1.12	3.27
Mammoth Copper M. Co...	.....	12	10.0	5.86	3.66	5.15
Average.....	.....	.....	7.22	4.76	2.17	5.52

*Estimated.

Timbering and Handling Ore in Stopes. Square-Set vs. Top-Slicing Methods.

There is often a question as to whether square setting or top slicing is more economical. For instance, in massive deposits of heavy sulphides the ore is easily drilled and breaks well and if overhand stoping with square sets is used the weight of the ore necessitates very heavy timbering. In order to compare the two methods and also to show cost data on timbering and handling ore in stopes, assume the methods applied to a block of ground 50 ft. square and assume that the development work is the same in each case. The sets to be 6 by 6 by 8 ft. In the top-slicing method a drift one set wide is started at the top of the raise and run to the boundary of the block and then across the end of the block. This will give 10 sets of drift. It will take about eight holes, each 4½ ft.

deep, to break a round, or 12 holes to break a set. All work in top slicing is breast work, and the drilling must be done with a drill mounted on a column. A mounted drill will drill slower and require more time for moving than a light, unmounted, stoping drill, which can be used in square-set mining. The cost of each set for the first 10 sets of drift in top slicing will be about as follows:

Drilling, labor, 2 shifts @ $3.50	$7.00
Drill repairs @ 0.30	0.60
Drill steel and sharpening @ 0.20	0.40
Air @ 1.00	2.00
Powder, 25 lb. @ 12½c.	3.00
Fuse, 60 ft. @ ¾c.	0.40
Caps, 12 @ 1c.	0.12
Cost for each set	$13.52

Each set contains 288 cu. ft. or approximately 28 tons; this makes the cost of drilling and blasting 48c. a ton. The distance to tram the ore to the raise will average 50 ft. and the cost for mucking and tramming may be estimated as follows: A man will shovel this ore at the rate of about 2½ tons an hour, or 1 ton in 24 min. A wheelbarrow will hold about ¼ ton and to wheel seven barrow loads 50 ft. will require from 10 to 15 min., say 11 min.; this makes the time consumed in shoveling and wheeling 1 ton 35 min. or 22c. a ton if shovelers' wages are 37½c. an hour. This is equal to 14 tons handled for each 8-hr. shift and a cost for each set of $6. Exclusive of the raise there will be a total of 63 sets in a slice, of which 10 sets are mined for 70c. a ton (48c. for drilling and blasting plus 22c. for mucking). The remaining 53 sets have two free faces to break to. The cost of breaking will be 30 per cent. less or 35c. a ton. The average distance from the raise will be less, which will reduce the cost of shoveling, say to 20c., which gives a cost of 55c. for the remaining 53 sets or an average cost of 57c. a ton to mine the whole slice. On one slice there will be required 72 posts, 64 caps, and 72 girts. A post 8 by 8 in. by 8 ft. will contain 42 ft. B.M. and costs 84c. if lumber is figured at $20 per M. Caps and girts 8 by 8 in. by 6 ft. will cost 64c. each, figured on the same basis. Framing the posts will cost about 10c. each by hand or 5c. each by machines, say 6c. Framing the caps and girts will cost about the same. This gives:

72 posts @ 90c.	$64.80
136 caps and girts @ 70c.	95.20
	$160.00

which is $2.55 a set. The cost of taking the timber into the stope and placing it will amount to from $1 to $2 a set, say $1.50. Other timber for lagging and blocking will cost about 85c. a set, which gives a total

for timbering of $4.90 a set or 17c. a ton. This added to the cost of breaking and mucking gives 74c. a ton.

To mine this same block of ground with square sets, it would be necessary to divide it into two stopes four sets wide and eight sets long. The ore has been assumed to be heavy and it would be impossible to carry a wide stope. As soon as a stope is mined it should be filled. For drilling, a light stoping drill can be used which, under the assumed conditions, will drill easily 50 per cent. more footage than the machine on the slice. One-third less explosives per ton will be required, so the reduction in cost of breaking will be at least 30 per cent. or to 30c. a ton. A flooring of plank will have to be laid on the top set and about 75 per cent. of the broken ore will not fall into the chutes but must be shoveled. A shoveler will handle 3 tons an hour at a cost of 12½c. a ton for 75 per cent. of the ore. This is equivalent to 9c. for the total tonnage. We have, therefore, reduced the cost of breaking and shoveling from 57c. in top slicing to 39c. in overhand stoping with square sets, a saving of 18c. a ton.

Consider the timbering. It is not necessary to go again into detail but first assume that the ground can be held up by square sets of 10 by 10 in. timber. A post 10 by 10 in. by 10 ft. long contains 66 ft. B.M. and at $20 per M. will cost $1.32. Since it is heavy timber and framed on both ends, the framing will cost about 12c., making its total cost $1.44. Caps and girts will cost $1 each plus 8c. for framing, or $1.08 total. This makes a total of $3.60 for timbers in the square set instead of $2.30 in the set used in top slicing, an increase of $1.30. There will also be an added cost of placing of about 50c., which makes a total additional cost for timbering in the square-set method of $1.80 per set or 6½c. per ton.

These figures show very plainly that the timbering used in square set must be very heavy to make the increased cost over top slicing greater than is the saving in breaking. Here the saving in the latter is 18c. but the increased cost of timber is only 6½c., leaving a balance of 11½c. in favor of square-set mining. This is in accord with experience, but when timbers will not support the ground and many braces are required, or filling must be resorted to, top slicing becomes cheaper. Filling will cost at least 20c. a ton and probably 50c. One mine that I have had in mind in preparing these figures has a cost of timbering and filling of more than $1 a ton. At the Esperanza mine, Mexico, in 1907, 35 ft. B.M. of timber were used for each ton of ore, which at $20 per M. is 70c., and that for timber alone. It must be remembered, however, that in ore that does not break easily the additional cost of breaking in the top-slice method becomes much greater.

I do not want to give the impression that these figures are to indicate the total cost of mining by the different methods. There are many other items such as superintendence and ventilation that enter into the cost of mining. There are also incidental expenses in breaking and timber-

ing, such as repairing, timbering chutes, and the like, the cost of which will enter into the total cost of any method, and the amount can be judged only by experience. *The figures given bring out only the relative costs of different operations in different methods under assumed conditions.* In my opinion, a few figures are necessary to aid in determining the merits of different methods or in reducing the costs of methods in use. With the figures, experience and judgment must be added. These are probably more necessary in mining than in any other business, because there are so many variable conditions.

Tramming

The costs of tramming as given in published reports nearly always include not only tramming, but loading also, either from chutes or from the floor of a drift. The following interesting data are from the *Engineering and Mining Journal* of Mar. 8, 1913.

"At the Elkton Consolidated Mining & Milling Co., Cripple Creek, Colo., the 1911 cost of tramming was 14.6c. per car of approximately 0.7-ton capacity. The South Utah Mines & Smelters, Newhouse, Utah, reports its tramming cost for the year ended June 30, 1912, at 15.76c. per ton of ore, which evidently includes the cost of handling waste removed. In Goldfield, Nev., tramming has averaged about 18c. per ton of ore produced from stopes and has ranged from about 14 to 25c. These figures are for actual tons trammed and do not include any shoveling in stopes. At the North Star mine, Grass Valley, Calif., observations show that a man pushes an 18-cu. ft. car about 150 ft. per minute and shovels about 3 tons per hour from a plat into car against 2 tons when shoveling off a rock bottom. According to this a shoveler's efficiency is increased about 50 per cent. by using a plat. The Wolverine Co., Houghton, Mich., reports tramming costs at 17.4c. per ton of ore, and the Wettlaufer-Lorain, Cobalt, Ont., 21c. per ton of ore."

Data on Tramming

Mine	State	Size Car Used	No. of Men	Length of Tram Ft.	Amount Trammed per Man-Hr.
			Shoveling from Rock Bottom		
Erie Consolidated	Calif.	1¼ ton	2	1,000	1.17 ton
Erie Consolidated	Calif.	1¼ ton	1	1,000	1.6 ton
Pittsburg-Silver Peak	Nev.	1.1 ton	1	700	1.52 ton
Cananea Consolidated	Mex.	16.8 cu. ft.	1	300	35.8 cu. ft.
			Shoveling from Plat		
Erie Consolidated	Calif.	1.0 ton	1	100	1.75 ton
Pittsburg-Silver Peak	Nev.	1.1 ton	1	1,000	1.575 ton
Cananea Consolidated	Mex.	16.8 cu. ft.	1	300	42.0 cu. ft.
Ohio Copper	Utah	20 cu. ft.	1	100	41.0 cu. ft.
			Loading from Chute		
Erie Consolidated	Calif.	1¼ ton	1	1,500	3.12 tons
Pittsburg-Silver Peak	Nev.	1.1 ton	1	700	6.19 tons
Cananea Consolidated	Mex.	16.8 cu. ft.	1	300	84.2 cu. ft.
Ohio Copper	Utah	20.0 cu. ft.	1	150	206 cu. ft.
Mother Lode	B. C.	2.15 tons	1	450	8.4 tons*

* Tramming with horses and locomotives.

If a man shovels from a plat at the rate of 2½ tons an hour and wages are 37½c. an hour the cost of shoveling is 15c. a ton. If a 1,000-lb. car is being used he will fill it in 12 min. If the tramming distance is 1,000 ft. and the trammer walks 200 ft. a minute the trip will require 10 min.; allowing 2 min. to dump, the tramming will take the same time and cost the same as the loading, that is 15c. a ton or a total of 30c. a ton. Three things are very evident: Tramming is an important item of cost in nearly every mine, the size of the car makes a decided difference, and the resistance to traction of the car itself and the grade and condition of the track are important factors. I remember seeing one stretch of track 1,000 ft. in length over which it took two men to push a car of 1,000 lb. capacity. Figuring as above this increased the cost of tramming 15c. a ton, and in fact it was even more because the two men could push the car but slowly.

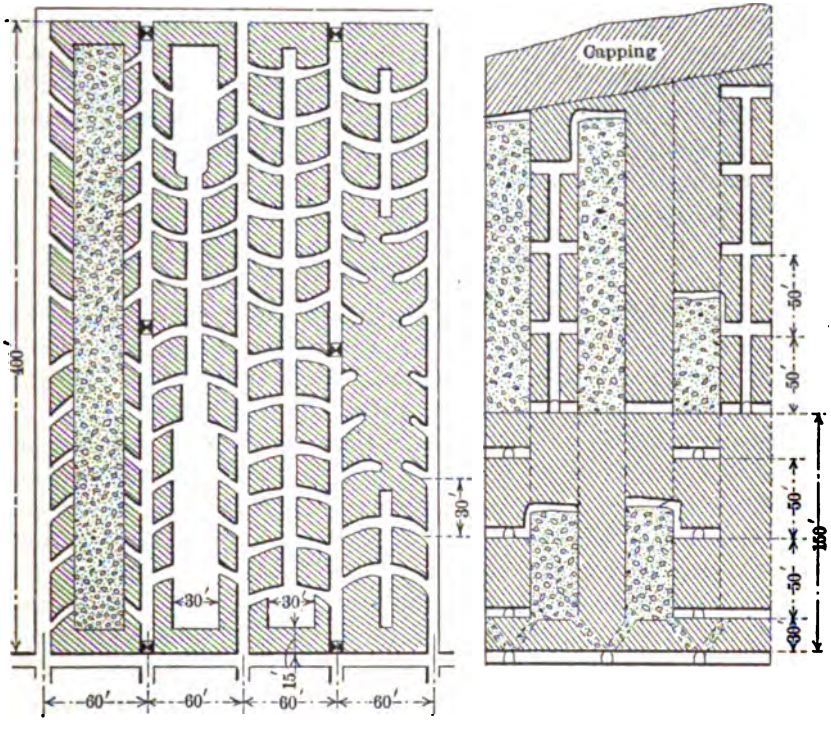
Henry Nagel, Superintendent of the Vindicator mine at Cripple Creek, has made some interesting observations in regard to tramming. He noted in one case that tramming on 800 ft. of level track cost 3c. a ton more than tramming on 800 ft. of track which had a grade of ½ per cent. in favor of the loads. In another case a man who had been shoveling and tramming in a drift with bad air, did 50 per cent. more work when a good circulation of air was secured. In his opinion the greatest efficiency is secured from a trammer when the track grade is such that he can ride the car when going in loaded. As to the latter assertion a few figures may show how it would not be true under all conditions. A car weighing 500 lb. and holding 1,200 lb. of ore will weigh 1,850 lb. when a 150-lb. man is riding. If it has good bearings and has a resistance to traction of only 20 lb. to the ton it will run on a 1 per cent. grade. If it has very poor bearings and has a friction of 60 lb. to the ton it will require a 3 per cent. grade for coasting. On this grade the empty car returning weighs 500 lb. and would require one-fourth of 120 lb. or 30 lb. force to push it. This is too much for the average man. The car with the good bearings would take only a push of 10 lb. to bring it back up the grade, which shows the importance of good bearings, as well as of good track.

Mining of the Massive Porphyry Copper Deposits

The past decade has witnessed the development and successful operation of a half dozen or more immense deposits of low-grade copper ore that occur in the Western States. In every case sufficient capital was available to carry out the development of the mines in the manner decided upon, and it is interesting to note the different methods that were employed.

Utah Copper and Boston Consolidated Mines

The first two "porphyry coppers" were the Utah Copper, and Boston Consolidated, at Bingham, Utah. The Utah Copper started out with a modification of the chute-caving method described in Crane's *Ore Mining Methods*, p. 141. The method was not satisfactory and was



PLAN END VIEW OF STOPES

FIG. 9.—STOPING SYSTEM, BOSTON MINE.

discarded for stopes and pillars, although very little mining is done by this method because the ore will be handled by steam shovels. The Boston Consolidated mine, now called the Boston mine of the Utah Copper Co., was originally laid out into stopes and pillars. Fig. 9 illustrates the method.[2] Above the main haulageway there is a 30-ft. pillar of ore to protect it. Above this the deposit is divided into a series of vertical stopes 30 ft. wide and 150 ft. high alternating with vertical pillars of the same dimensions. Above these comes another and similar

[2] Figure taken from article by C. T. Rice in *Mines and Methods*, September, 1910, and copied in *Mining Without Timber*, p. 172.

series of stopes and pillars, but the pillars in the upper series come immediately above the stopes in the series below. The plan was to work all the stopes on both levels as shrinkage stopes. When the stopes were all completed drawing would commence from the bottom of the lower stopes. The upper pillars being over stopes below would settle and crumble so that they would pass down with the ore from the upper stopes when the lower pillars were weakened by blasting. It will be noted that in this method the capping must cave and follow down on top of the broken ore. Just what alterations in the plan of working would have been necessary if the method had remained in use it is difficult to state. The stopes are still being worked but only the excess ore is drawn off; the remaining ore is left to be mined by steam shovels after the capping is removed. The width of stopes has been reduced from 30 to 18 ft., and the pillars increased to 42 ft.

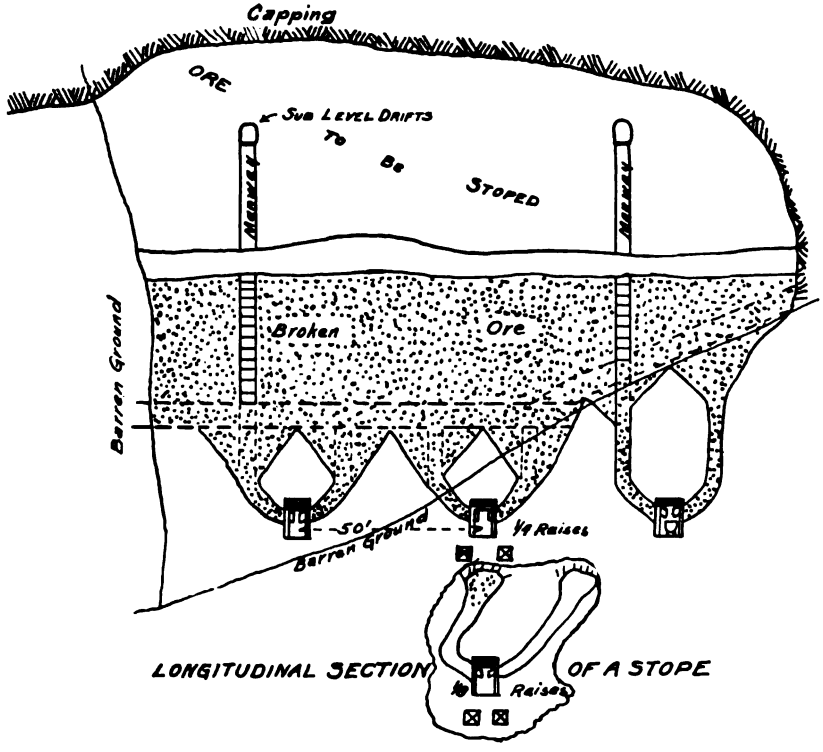
Ray Consolidated Mine

The Ray Consolidated Copper Co. at Ray, Ariz., is controlled by the same interests as the Utah Copper Co. and when their system of mining was laid out they no doubt profited by the earlier experience at Bingham. The orebody is flat and averages a little more than 100 ft. thick. The main haulageways are at the bottom of the ore and above them the ore is divided into vertical stopes only 15 ft. wide leaving 10-ft. pillars between, as shown in Fig. 10.[3] The stopes are worked as shrinkage stopes to the top of the orebody, then the bottoms of the pillars are blasted and drawing is commenced from under both stopes and pillars. The pillars, being undercut, settle and break up as the ore is drawn. By taking care to draw equally from all the chutes under a block of ground the capping will cave and follow down uniformly on top of the ore and there will be little mixing of the two. The Ray system, from all reports, has been eminently successful. Stopes 15 ft. in width are wide enough so that a large tonnage will break compared to the footage drilled, and all the advantage is had of the shrinkage method of stoping. After the ore in the stopes is broken, the pillars, which contain more than one-third of the total ore, are available for extraction with only a slight expense for drilling and blasting. The cost of mining for the last quarter for which reports have been published was 71c. a ton. This includes a proportion of all general and fixed charges, but does not include an allowance of 12½c. a ton for the retirement of mine-development suspense account.

Miami Mine

The orebody of the Miami Copper Co. is much deeper than that at Ray. In horizontal cross-section it is roughly circular, about 1,000 ft.

[3] Abstract of article by L. A. Blackner, *Mining and Scientific Press*, Jan. 3, 1914.



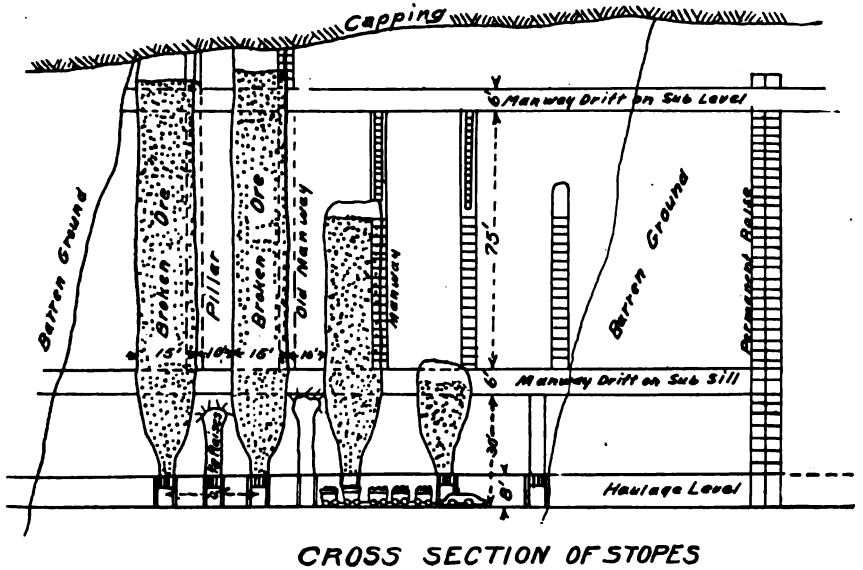


FIG. 10.—SECTIONS OF STOPES, RAY CONSOLIDATED MINE.

across. A tongue of waste rock intrudes into the ore from one side. The irregularities in the orebody under the capping are first worked by square-set stopes and timber is placed to form a mat between ore and waste. Through the main body of ore sublevels are spaced 25 ft. apart vertically as shown in Fig. 11[4] and on these sublevels drifts and cross-drifts are driven each way 50 ft. apart. The main body of the ore is to be mined with shrinkage stopes and pillars similar to the Ray method except that

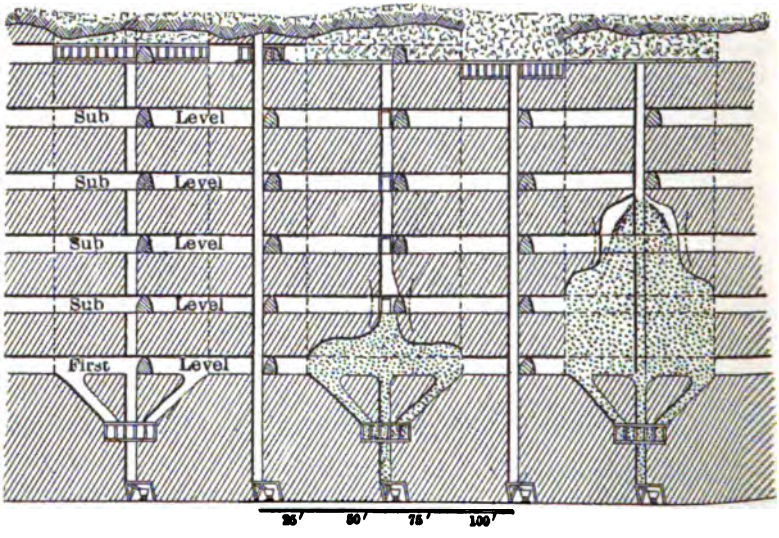


FIG. 11.—CROSS-SECTION OF STOPE, MIAMI MINE.

at Miami the stopes were planned to be 60 ft. wide and the pillars 40 ft. In working the stopes the miners do not work on top of the broken ore but approach the stopes through the drifts on the sublevels and break the ore into the stopes by the method of sublevel stoping previously described. The ore breaks readily into small pieces and obviously a 60-ft. shrinkage stope would be very dangerous if worked in the usual way, but by the sublevel method the danger is eliminated. Even under these conditions the last report states that it has been found advisable to make the stopes narrower. After the shrinkage stopes are carried up to the mat, which is next to the capping, the pillars are worked from the top down by top slicing. As this is done the ore from the stopes is drawn and a uniform settlement of the capping results.

The cost of mining at Miami has been about $1.20 a ton. Last year a premature crushing of square sets under the capping started a cave

[4] R. L. Herrick in *Mines and Minerals*, July, 1910; *Mining Without Timber*, p. 167.

which extended through into the stopes below. The repair of this damage caused an increase in the cost for 1913. The Miami orebody is of higher grade than that at Ray and the Miami method will save a greater percentage of clean ore. The cost per ton is greater and there is a question as to how much the cost can be reduced in the future. To date, a large amount of ore has been taken from the square-set stopes, an expensive operation, but, on the other hand, in the future there will be an increased amount of ore to be mined from the pillars, which is also expensive. It would be interesting to know how much of the difference in cost between the methods at Ray and Miami is due to the difference in cost of labor. At Miami the miners receive $3.50 a day and up. At Ray mostly Mexicans are employed and they are paid, I believe, about $2.50 a day and up.

Inspiration Mine

The Inspiration mine, adjoining the Miami, has not yet begun to produce on a large scale, but it has been developed, in part at least, with the idea of using block caving. The method was described by Claude T. Rice in *Mines and Methods*, June, 1909, and is similar to the block caving illustrated earlier in this paper. The individual blocks were to be 75 by 200 by 200 ft. high and were to be isolated by stopes on the sides and breast stopes between ore and capping. The pillars under the blocks instead of being as shown in Fig. 7 were to be very narrow, 75 ft. long and 75 ft. high. Between the pillars narrow shrinkage stopes filled with ore would prevent the pillars from early crushing. This method, I believe, has never been given a trial and now is to be replaced by the method of block caving which has been so eminently successful at the Ohio Copper mine at Bingham. This method is worthy of a fuller description. The following is abstracted from a description of the method by Clarence G. Bamberger in the *Engineering and Mining Journal* of Apr. 6, 1912.

Ohio Copper Mine

"In *résumé* the conditions are: An orebody opened 400 ft. in width, 450 ft. in length, 1,300 ft. in depth, dipping from the horizontal at an angle of 50°. The entire mass being a broken shattered quartzite containing copper, chiefly in the form of chalcopyrite disseminated throughout, inclosed by foot and hanging wall of the same formation with boundaries not clearly defined.

"The preparation of this ground for a caving system of mining has been carried out as shown in Fig. 12. An incline shaft was sunk in the foot wall on the dip of the vein connecting with the different levels for the transportation of men and supplies. Two main ore chutes No. 1 and No. 2 were driven in the foot wall on an angle slightly greater than the dip of the vein. . . . . These chutes connect by diagonal raises through the foot wall, with the different levels where extraction is taking place and

deliver into bins of large capacity at the Mascotte tunnel or main haulage adit which in turn leads to the concentrator situated at the portal about 3 miles distant.

"Lateral main levels were driven across the orebody at the 100-, 300-, 400-, 500-,

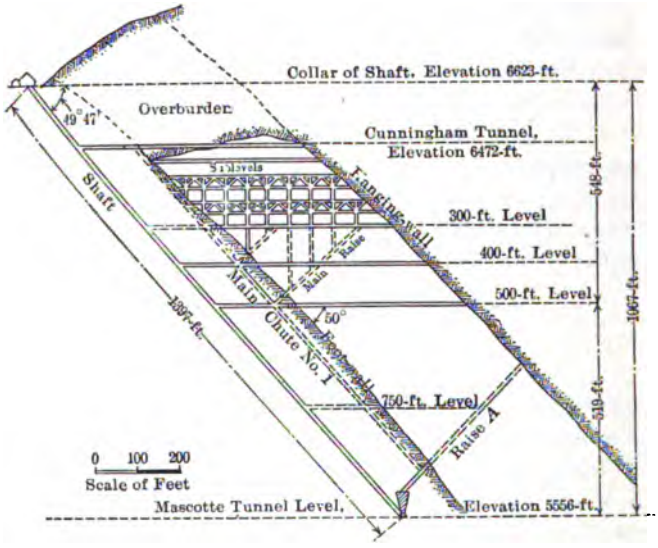


FIG. 12.—CROSS-SECTION, OHIO COPPER MINE.

750-, and 1,000-ft. or haulage level. By a network of crosscuts and drifts these different levels have been cut up into blocks 200 ft. square which in mining will again be subdivided. Between the several levels the ground has been blocked out into sublevels

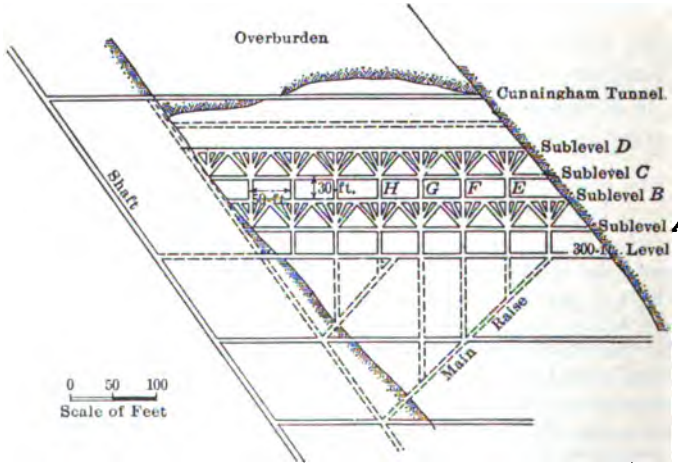


FIG. 13.—CROSS-SECTION, OHIO COPPER MINE.

30 ft. apart and again subdivided by crosscuts, drifts and raises into blocks approximately 30 x 50 x 25 ft.

"The actual procedure of extraction can be clearly understood by reference to Fig. 13 which shows in detail the sublevels above mentioned. Thus we have for example the four sublevels, *A*, *B*, *C*, and *D*. Raises are driven from the crosscuts and drifts on the 300-ft. level through sublevels *A* and *B*. From these vertical raises at the various points *E*, *F*, *G*, and *H* raises are run into sublevel *C*, radiating like fingers from the palm of the outstretched hand. At the breast of these several raises sublevel *D* is blasted down, the ore falling by these leads into the cross inclined chutes connecting with the main ore chute, which delivers into bins at the loading station on the haulage-tunnel level.

"The same procedure is carried out on sublevel *B* which in turn is blasted down; at the same time sublevel *C*, which has already been cut up by this finger-like network of raises, comes with it as the solid ground below is blasted down. Thus each alternate sublevel is cut up by the numerous raises and the corresponding sublevel above and below is blasted down.

"The actual cost per ton for the different phases of the operation is shown in the accompanying table which is an average covering a period of 31 days in October, 1911, during which period 56,311 tons were mined.

Typical Daily Labor Report. Ohio Copper Co.

Description of Labor	Number	Rate	Amount
Superintendent	1	$11.66	$11.66
Foremen	2	5.33	10.66
Shift bosses	3	4.00	12.00
Timekeeper	1	3.00	3.00
Sampler	1	3.00	3.00
Mine engineer	1	5.00	5.00
Hoist engineer	2	3.25	6.50
Stationary engineer	2	3.50	7.00
Chainman	1	3.50	3.50
Tool sharpener	2	3.50	7.00
Machine men	21	3.25	68.25
Machine men	8	3.00	24.00
Muckers	23	2.50	57.50
Muckers	10	2.75	27.50
Loaders	2	3.25	6.50
Loaders	3	3.00	9.00
Chute tappers	3	3.00	9.00
Nippers	2	2.50	5.00
Blacksmith	1	4.25	4.25
Blacksmith helper	1	3.25	3.25
Carpenter	1	4.00	4.00
Carpenter's helper	1	3.25	3.25
Trackmen	1	3.00	3.00
Trackman's helper	1	2.50	2.50
Pipeman	1	3.25	3.25
Pipeman's helper	1	2.75	2.75
Timbermen	7	3.25	22.75
Timbermen's helpers	7	2.75	19.25
Total	110		$344.32

Total labor charge for Oct., 1911	$11,582.78
Total stores consumed for period	3,743.82
Total power consumed for period	476.00
Total operating expense for period	$15,802.60
Total development and equipment for period	3,429.00
Total actual operating expense for period	$12,373.60
Average number of feet driven per day	31
Average number of feet raised per day	42
Average tons mined per day	1,817
Total number of feet driven for period	961
Total number of feet raised for period	1,302
Total tonnage mined for period	56,311

All calculations based on net weight.

"From the above data several interesting conclusions can be drawn. Considering the whole working force of 110 men as producing the ore, approximately 17 tons of ore are delivered to bins per day per man. Considering, however, only the men who are actually breaking the ore, approximately 63 tons of ore are delivered to bins per day per man. From the 'total operating expense for period' and 'total tonnage mined for period,' the cost per ton delivered to bins is shown to be 28.06c., which figure includes development and equipment charges. From 'total actual operating expense' and 'total tonnage mined for period,' the actual cost per ton for mining delivered to bins is shown to be 21.97c."

This method of mining is similar to the ideal case of block caving already described, only instead of having to shovel the ore after it is caved a large number of branching raises are brought up from underneath and the ore runs into these and on down to bins at the bottom of the mine without any handling. The raises contain chutes and the ore is drawn evenly from a large area so that the capping will follow down without mixing too much with the ore. While considerable ore is lost the cheapness of the method is remarkable. I have in my possession the figures for April, 1913, almost two years after the above figures were taken, and the cost of mining including development and equipment is reduced from 28.1c. to 22.2c.

Development

The extensive development necessary for the successful operation of some methods of mining naturally brings to mind the question of what this work costs. In the appendix, I have placed a collection of figures on the cost of development as published in the reports of mining companies Easily driven drifts and crosscuts will cost $3.50 to $4.50 a foot. Average-sized drifts in hard rock with a small amount of timbering cost $6 to $7 and under unfavorable conditions the cost will be $10 or more a foot. In a 5 by 7 ft. drift costing $6.50 a foot the labor for drilling and blast-

ing will be about $2.15, labor for shoveling and tramming about $1.20, air about 30c. a foot, explosives about $1.25, and miscellaneous 60c. a foot. Raises are as a rule less expensive than drifts and, in very favorable ground, can be driven for $1.50 to $2.50 a foot. The average 4 by 5 ft. raise in hard rock will cost $4 to $5 and raises difficult to drive or requiring close timbering will cost $8 a foot and up. Winzes are not driven often but when they are they cost from $10 a foot up.

Space permits of a consideration of only a few methods from the point of view of development. At Kalgoorlie (J. Cheffirs, *Transactions Australian Institute of Mining Engineers*, vol. xiii) in a block of ore 500 ft. long and 200 ft. high (the distance between levels), rill stoping required four raises and shrinkage stoping required one; 800 ft. of raises cost about $5 a foot or $4,000. If we assume the vein to average 8 ft. wide a block of ore would contain about 60,000 tons, which means a cost for development of almost 7c. a ton. With shrinkage stoping this cost would be less than 2c. a ton. Of these methods touched upon in the preceding pages the Miami mine has the greatest footage of development work. The scheme of development allows a good many different methods of mining to be used without any changes; it also permits a large tonnage to be produced from a limited area. A block of ore 50 by 50 by 50 ft. contains approximately 50 ft. of raise and 200 ft. of drifts and crosscuts. Estimating raises at $3 a foot and drifts at $5 a foot gives a total of $1,150 for the block. It contains about 10,000 tons of ore, which would make 11½c. a ton as the cost for development; this excludes the cost of shafts, stations, and similar requirements.

At Ray 12½c. a ton is allowed for the retirement of mine-development suspense account but just what this covers is not stated. It evidently covers more than development for stoping.

The method used at the Ohio Copper mine requires considerable development of which a large percentage is raising. Due to the fact that many of these raises were inclined, so that a miner could work at the face without using timber, and that all the ore, after being blasted, runs by gravity to the bins, the cost of the raises was only at the rate of 50 or 60c. for each ton of ore mined in driving them.

APPENDIX

Mining Costs

Montana-Tonopah Mine. (*Mining and Scientific Press*, Mar. 22, 1913)

Silicified and mineralized veins in altered andesite, 3 to 5 ft. wide, steeply pitching. Sometimes thrown over by faults through the stopes. Overhand stoping.

Year ended Aug. 31, 1912; 53,874 tons mined.

Labor:	
Ore breaking	$0.579
Mine machines	0.031
Hoisting and dumping	0.197
Boilers	0.034
Shoveling and sorting	0.675
Tramming	0.224
Timbering	0.209
Tool sharpening	0.026
Surveying	0.028
Foreman and bosses	0.078
Sampling	0.016
Storekeeper	0.011
Assaying	0.016
Watchman	0.011
Superintendence	0.046
Maintenance and repairs	0.001
Total labor	$2.182

Supplies:	
Water	$0.005
Ore breaking	0.292
Compressed air	0.111
Hoisting and dumping	0.059
Hoisting (elec. power)	0.129
Timbering	0.197
Total supplies	$0.793
Total mine cost	$2.975

Tonopah-Belmont. (*Engineering and Mining Journal*, May 11, 1912)

Year ended Feb. 29, 1912; 182,000 tons dry ore and waste; 152,550 tons dry crude; 115,560 tons dry, sorted.

Fracture zones in andesite and rhyolite-dacite, filled with quartz. Overhand stoping.

	Per Ton Dry, Sorted, Cents
Development:	
Miners	47.4
Muckers and trammers	22.2
Timbermen and helpers	8.4
	78.0
Stoping:	
Miners	44.5
Shovelers	33.9
Trammers	19.2
Timbermen and helpers	97.8
Filling	4.2
Piston-drill repairs and maintenance	5.0
Stoping-drill repairs and maintenance	2.9
Steel and sharpening	7.1
Explosives	28.6
Hoisting to surface	30.9
Auxiliary hoisting	9.4
Ore sorting and loading	27.3

Sampling and assaying	4.7	
Surveying	5.6	
Supt. and shift bosses	13.7	
Mine office	12.9	
Surface and plant	16.2	
Lighting	4.6	
Heating	4.1	
Drayage	6.1	
Maintenance and repairs to buildings	2.5	
Maintenance and repairs to machines and machine tools	2.3	
Maintenance and repairs to pipe lines and tanks	1.6	
Maintenance and repairs to railroad spurs	0.8	
Maintenance and repairs to pole lines	0.2	
Pumping	6.2	
Ventilation	1.7	
	394.0	
		$3.94
		0.78
		$4.72
Administration, taxes, safety, and depreciation		0.719
Grand total		$5.439

West End, Tonopah. (*Mining and Scientific Press*, Aug. 16, 1913)

Superintendent and foreman	$0.135
Breaking	0.802
Timbering	0.090
Tramming	0.372
Hoist, etc	0.200
Ore loading	0.233
Ore sorting	0.366
Assaying, sampling, surveying	0.091
Surface, ore dump, drayage	0.195
Development	0.862
General expense	0.554
Miscellaneous	0.262
Total	$4.162

Bunker Hill and Sullivan. (*Engineering and Mining Journal*, June 14, 1913)

Large replacement deposits of sulphide ores in quartzite. Overhand stoping with stulls or square sets.

	Labor	Supplies
Superintendents, blacksmiths, and supply men	$0.177	.....
Timbering	0.086	$0.215
Miners	0.367	.....
Carmen	0.052	.....
Shovelers	0.366	.....
Power	0.027	0.045
Repairs	0.026	.....
Explosives	.....	0.075
Illuminants	.....	0.020
Lubricants	.....	0.003
Iron and steel	.....	0.012
Miscellaneous supplies	.....	0.035
Wood	.....	0.034
Stable	.....	0.001
Total	$1.101	$0.440

Grand total $1.541

Stewart Mining Co., Cœur d'Alêne. (*Mining and Scientific Press*, Apr. 26, 1913)

Cost of mining and development $2.15

Ferreria Mine, Rand, South Africa. (*Mines and Minerals*, March, 1911)

Overhand rill stoping, shrinkage.

Tons for each machine shift	10.48
Pounds of explosives per ton, 0.57, cost	16c.
Total cost of stoping, 77c., plus 25c. for timbering,	$1.02
Stoping on contract, per ton	55c.

Brakpan Mine, Johannesburg. (*Mining and Scientific Press*, May 17, 1913)

Presumably underhand stoping.

Mining:

Stoping	$1.11
Timbering and packing	0 24
Shoveling and tramming mine and dump	0.66
Transport, underground	0.11
Transport, surface	0.01
Hoisting	0.20
Pumping	0.16
Other charges	0.17
Development	0.36
Total	$3.02

Rand Mining Costs. (E. M. Weston: *Engineering and Mining Journal*, Feb. 8, 1913)

Presumably all underhand stoping.

New Kleinfontein Mine.—With hand drilling each native breaks 1.2 tons a shift at a cost of $1.03. Of this amount the white labor cost $0.16, the native labor nearly three times as much, while food, etc., for the latter costs 22c. to 24c. per ton. Owing to easy nature of ground, explosives cost only a little over 14c. a ton. The average footage drilled per native each shift is 40 in. and the stoping width 4½ ft.

At the same mine large machines broke 10 tons a shift in a 70-in. stope at a cost of 97c. a ton broken; of this 97c., white drillers cost 23c., natives 11½ c., while their food, etc., cost 30c. Explosives cost 21c. a ton.

One of the Deep-Level Mines.—Hammer boys drilled 33 in. per shift and broke ½ ton in a 44-in. vein at a cost of $2.73 a ton. Explosives cost 30.3c. a ton. Large machines broke 7.2 tons a shift in a 55-in. vein at a cost of $1.87 a ton. Explosives cost 36.3c. a ton. Small machines broke 3.7 tons in a 4-ft. vein at a cost of $2.43 a ton. Explosives cost 40.5c. a ton.

One of the Large Outcrop Mines of the Central Rand.—Each native drilled 48 in. and broke 1.5 tons a shift in a 63-in. vein at a cost of $1.22 a ton. Of this explosives cost 18.2c. Large machines broke 19.5 tons a shift in an 83-in. vein at a cost of 77c. Of this, explosives cost 20.2c.

The above costs are for stoping only. The total mining costs ran from $2.25 to $5 a ton.

Portland Gold Mining Co., Cripple Creek, Colo.

Overhand stoping with stulls and square sets.

	Per Ton Broken	
Tramming	$0.17	
Hoisting	0.10	
Sorting	0.14	equal to 30c. per ton of ore through ore house.
Assaying, engineering, superintendence, and general	0.30	
Stoping	1.63	
Total	$2.43	

About 44 per cent. of total ore broken is trammed; 29 per cent. of ore trammed is sorted, equal to 12.7 per cent. of total.

The costs for the year 1906 at the Portland mine as given in *Ore Mining Methods*, by W. R. Crane, were as follows:

	Cost per Ton
Labor	$1.142
Machines	0.270
Tramming	0.029
Explosives	0.380
Hoisting	0.230
Supplies	0.036
Superintendence, assaying, etc.	0.450
Total	$2.537

The labor cost was subdivided as follows:

Machine men	$0.4761
Trammers	0.3214
Pipe and track men	0.0357
Timbermen	0.1666
Timber helpers	0.1428
Total	$1.142

Temiskaming Mining Co., Cobalt, Canada. (*Engineering and Mining Journal*, May 25, 1912)

Narrow veins in schist. Overhand stoping.

Mining and timbering $1.85, of which labor was 66.5 per cent. and power 13.5 per cent. Dynamite cost 33.7c. a ton, fuse 2.6c., candles 3c., drill repairs 21c., steel 14.7c.

Hollinger Mine, Porcupine, Canada. (*Mining and Scientific Press*, May 3, 1913)

Country rock, fine-grained compact schist. Veins 2 to 6 ft. wide, containing quartz in stringers and bunches. Overhand stoping.

General and superintendence	$0.179
Diamond drilling	0.027
Stoping and driving	1.969
Timbering stopes	0.219
Tramming	0.551
Drainage and pipes	0.092
Hoisting	0.170
Dumping	0.063
Drill steel	0.298
Assaying, sampling, and surveying	0.064
Change house and lights	0.013
Handling explosives	0.025
Handling waste	0.016
	$3.686

In the Kalgoorlie district, Western Australia, the ore occurs in strong veins in quartz-dolerite. The width of the veins varies but averages about 12 ft. The stoping methods employed are overhand rill stoping with filling, flat-back stoping, and shrinkage stoping. The following costs are of mines in this district.

Great Boulder Perseverance. (*Engineering and Mining Journal*, May 25, 1912)

Average stoping width 12.94 ft.

Wages and contracts	$0.92
Explosives	0.156
Drill parts and air lines	0.0434
Candles	0.0182
Air for drilling	0.1056
Not specified	0.4682
Total cost	$1.71

Ivanhoe Mine. (*Mining and Scientific Press*, May 24, 1913)

Ore breaking	$1.38
Filling stopes	0.27
Tramming and hoisting	0.58
Development	0.52

Kalgurli Mine. (*Mining and Scientific Press*, Mar. 8, 1913)

Labor:	
Superintendence	$0.04
Breaking ore	0.66
Timbering stopes and chutes	0.04
Loading and tramming	0.42
Filling stopes	0.16
Tool sharpening, etc	0.05
Sundries	0.02
Total labor	$1.39
Stores:	
Tools, steel, drill renewals	$0.03
Candles	0.02
Explosives	0.18
Timber	0.04
Assays	0.01
Sundries	0.01
Total stores	0.29
Hoisting, machine drills	0.24
Grand total	$1.92

Oriental Consolidated, Unsan, Korea. (*Mining and Scientific Press*, Apr. 30, 1910)

Mining timbers	$0.254	Picks, shovels, hammers	$0.009
Firewood	0.133	Miscellaneous (rope, pipe, cars, etc.)	0.046
Lumber	0.105		
Dynamite	0.097	Total supplies	$0.823
Candles	0.094	Assays	0.006
Fuse	0.094	White labor	0.117
Detonators	0.011	Korean labor	0.612
Charcoal	0.013	Outside expense, shops, stables, etc	0.030
Lubricants	0.010		
Drill steel	0.011	Total	$1.590
Bar, sheet, track iron	0.010		

Consolidated Mercur Gold Mining Co., Mercur, Utah. (*Mining and Scientific Press*, Sept. 25, 1909)

Fifteen to 70 ft. vein of soft ore in hard cherty limestone. Dip 10° to 30°. Sublevel caving, with sublevels 14 ft. apart.

Total cost of mining $1.53.

Alaska-Treadwell

From the costs of 1912:

Machine drillers	$0.143	Iron and steel	0.003
Laborers, powdermen, etc.	0.128	Lumber and timber	0.003
Foremen	0.009	Compressed air	0.018
Blacksmiths	0.006	Power	0.007
Machinists, carpenters, timbermen	0.005	Mechanical repairs	0.002
		Blacksmith shop	0.005
Powder	0.146	Miscellaneous	0.041
Fuse and caps	0.016		
Candles	0.006	Total	$0.544
Machine drill supplies	0.006		

Cost of stoping only.

Anaconda Co., Butte, Mont.

Cost of mining by overhand stoping with square sets and filling was $3.77 in 1911.

Braden Copper Mine, South America. (*Mining and Scientific Press*, Dec. 4, 1909)

Ore breaking including superintendence and general charges was 41c. Machine drilling with 2¼-in. drill was at the rate of 31 ft. per man-day at a cost of 2.025c. a foot. Hand drilling was done at the rate of 13.8 ft. per man-day at a cost of 8.2c.

Cost of Development

Bunker Hill and Sullivan, Cœur d'Alêne. (*Engineering and Mining Journal*, June 14, 1913)

11,050 ft. of vertical and horizontal development:

	Per Foot	Per Cent.
Foremen, blacksmiths, etc.	$0.312	4.4
Miners	2.500	35.0
Shovelers	1.650	23.1
Explosives	0.990	13.8
Timber and lagging	0.400	5.6
Power, labor and supplies	0.524	7.33
Not specified	0.774	10.8
Total	$7.15	100.0

Portland Gold Mining Co., Cripple Creek, Colo.

Cost of drifts or levels, $6.12; crosscuts, $6.23; winzes and raises, $8.60. In the *Cost of Mining* by Finlay, p. 380, the cost of 896 ft. of 5 by 7 ft. drifts is given as follows:

	Per Foot		Other costs:
Tramming	$1.00	Use of machines, air, etc.	$0.97
Pipe and trackmen	0.14	Repairs, cars, etc	0.08
Machine men	1.88	Explosives	1.43
		Hoisting	0.46
Total labor	$3.02	General expense, surveying, assaying, bosses	0.58
		Grand total	$6.20

Montana-Tonapah Mining Co. (*Mining and Scientific Press*, Oct. 9, 1909)

Drifting, $6.56 a foot; crosscuts, $5.44; raises, $4.65; winzes, $11.92. Cost of development subdivided as follows:

Labor:	Per Cent.
Breaking	27.10
Timbering	4.38
Hoisting and dumping	6.32
Foremen and shift bosses	1.75
Blacksmith, sharpening	1.75
Shoveling and tramming	22.90
Surveying	1.35
Watchman	0.52
Storekeeper and timekeeper	0.35
Diamond drill hole	4.38
Supplies:	
Breaking	21.20
Timbering	1.23
Hoisting and dumping	2.86
Hoisting, electric power	3.56

At a Western Gold Mine 302 ft. of development work took 238 miner's shifts, 122 shifts of muckers and trammers, and 105 machine-drill shifts. Explosives cost $1.41 a foot and air for drills 25c. a foot or 75c. for each machine-drill shift.

At the Standard mine, Bodie, Cal., drifts cost $3.07 a ft. with an average of 1.3 ft. a shift, and raises cost the same. At the *Commercial mine at Bingham, Utah*, in 1912, drifts cost from $5.83 to $8.13 a ft., and raises cost $5.56. At the *Nevada Hills mine, Fairview, Nev.*, drifts and crosscuts cost $9.75 a foot, raises $12.50, and winzes $30. At the *Great Boulder Perseverance, Kalgoorlie*, drifts cost $12.90 a foot, and crosscuts $13.80. At the *Braden Copper Co., South America*, development cost $3.54 a foot.

At Cananea in hard quartz a 4½ by 6½ ft. drift took 7.8 lb. of powder for each foot, and a 6 by 11 ft. raise took 8.3 lb. At the *Pittsburg-Silver Peak* a 4½ by 5 ft. raise took 9 lb. powder for each foot, and at the *Erie Consolidated* in slate and quartz a 5 by 7 ft. raise took 6.65 lb. In drifts driven by hand work in medium-hard ground the cost was $4.50 to $5.50 a foot of which the labor was 77 to 85 per cent. and explosives 15 to 23 per cent. The rate of advance was 1 to 1½ ft. a shift.

Mining Claims within the National Forests

BY E. D. GARDNER, MISSOULA, MONT.

(Salt Lake Meeting, August, 1914)

WHEN the National Forests were created, all lands embraced in their boundaries were exempted from all forms of entry, except mineral claims. Later, by Act of June 11, 1906, and as amended by Act of Aug. 10, 1912, provision was made allowing homesteads on areas chiefly valuable for agriculture and not necessary for the public use. Patents to mining claims in the Forests are granted the same as on land not in the Forests, irrespective of timber or other values.

It is the policy to encourage the highest use of the Forests. As a business proposition, it is a good policy to have mines developed in the Forests. Their development in remote or inaccessible regions causes the country to be opened up and, in addition to the local market which the mines create, means of transportation are afforded to outside markets.

No examination of mining claims is made until patent has been applied for, unless they are actively interfering with the administration of the Forest. In this case, an examination is made, and if there is evidence that the claims are being held in good faith they are in no way interfered with. Timber is never sold from a mining claim except with the consent of the claimant, or unless it is clearly shown, after an examination, that the claim is abandoned. In the case of fire-killed timber, which rapidly deteriorates by standing, a contract bond arrangement can be made for the sale of the timber which protects the interests of both the mining claimant and the government.

The promiscuous locating of mining claims has in the past caused considerable trouble. It has been, and still is, the custom in some parts of the West to locate a mining claim when the surface rights of any particular area are desired, irrespective of the mineral character of the ground. This has been done to hold power sites, town sites, summer residences, springs, etc. When the ground is desired for any of these purposes, what is known as a Special Use Permit will be issued at a nominal sum, to legalize the occupancy. In most instances, it is far cheaper to take out one of these permits than to do the annual assessment work on a mining claim, but, of course, patent to the ground cannot be obtained in this manner.

If mineral were discovered on the land covered by a Special Use Permit, it would ordinarily be the policy to cancel the permit if it were clearly

shown that the land in question was more valuable for mining than for the purpose for which the permit was granted. The burden of proof in such a case must be on the mining claimant. For example, in the case of a permit for a large storage reservoir, it would take a very good mineral showing to justify its cancellation. If a valid mineral claim occupied ground previous to the application for a Special Use Permit, none would be granted without the consent of the mining claimant.

An unpatented mining claim in a National Forest cannot be put to any use inconsistent with the development of the claim, without a permit, which would be charged for at the regular rate.

In the past, before the government inspected ground prior to issuing patent, many valuable power sites, town sites and timber areas have been patented as mining claims when the ground was in no respect mineral.

It has often been stated that the patenting of mining claims is seriously delayed if they are within a National Forest. Out of over 150 claims examined during each of the years 1912 and 1913 in Montana and Idaho, north of Salmon river, only 17½ per cent. were contested. The records show no delay longer than three months except where patent was applied for during a time of year when field examinations could not be made on account of climatic conditions.

Patenting Claims

Very few claims that come up to patent comply with all the Federal and State laws in every particular. Most of this non-compliance, however, is due to ignorance of the law. One reason for this ignorance is that the requirements for staking and patenting mining claims are not the same in all States. Most prospectors and miners operate in more than one State and many times confuse the various regulations.

In accordance with the mining laws, patent on a mining claim should be issued when the claimants show: first, good faith; second, the requisite amount of work done; and third, a mineral discovery made. No protest is made against issuance of patent by the Forest Service when the location is not in accordance with all of the State laws, or for any non-compliance with the minor technicalities of the Federal law. For instance, an application for patent would not be protested for the reason of the vein not being in the middle of the claim or for striking across it.

Good Faith.—On a claim that is chiefly valuable for mining, good faith is assumed to have been shown when the requisite amount of work has been performed and a mineral discovery made. When it is evident that the ground is more valuable for other purposes than for mining, a merely technical compliance with the requirements of the law does not suffice to make clear the good faith of the applicant.

Development Work.—Five hundred dollars is required to be expended

on or for the benefit of a claim before patent is granted. When a claim is examined for patent the value of the work is estimated as closely as possible from the facts at hand, but when any doubt exists the claimant is always given the benefit of the doubt. When work is done on another claim for the benefit of the one in question, the Service does not take a narrow and technical view of what would tend to develop it.

Mineral Discovery.—Application is often made for patent on claims held in apparent good faith but on which no mineral discoveries have been made. Such claims are the so-called protection, or side, claims, and the odd claims in large groups. These protection claims in many cases are really necessary to the group to avoid possible future trouble with later locators and for tunnel sites, etc.; but, no matter how necessary they are, the law requires that a mineral discovery must be made on each claim before a valid location can be made. Most claims which are contested by the government are of this character.

When, after a sufficient expenditure has been made, patent is desired on a group that includes claims which do not contain mineral discoveries, the common improvement work should be credited to all of the locations and application made only for those which meet the requirements of the law. Patent can be asked for the protection claims under a separate and later application after mineral discoveries have been made. As long as the assessment work is kept up, possession of such claims can be retained.

There is a great difference of opinion as to what constitutes a mineral discovery. According to late decisions of the Commissioner of the General Land Office in the East Tintic case of June 15, 1912, and again in the Jefferson Montana Copper Mining Co. case of Sept. 5, 1912, it was held that the following elements are necessary to constitute a valid discovery upon a lode mining claim:

1. There must be a vein or lode of quartz or other rock in place.

2. The quartz or other rock in place must carry gold or some other valuable mineral deposit.

3. The two preceding elements, when taken together, must be such as to warrant a prudent man in the expenditure of his time and money in the effort to develop a valuable mine.

In passing on a mining claim, the Forest Service Mineral Examiner takes into consideration the general geological conditions of the locality. For instance, in the Cœur d' Alêne and the Butte districts the ore or mineral values do not usually outcrop at the surface, and the presence of a well-defined vein with the characteristic iron-stained vein matter upon a claim, considering the experience gained on other veins in the districts, would afford a reasonable expectation of finding ore at depth.

In determining a mineral discovery on a placer claim, the values found are considered together with the extent of the gravel and mining condi-

tions. A very small value to the pan of gravel would be considered as a mineral discovery where conditions were such that there was any possibility that the ground would ever be mined, but a few isolated colors found on a claim would not in themselves constitute a mineral discovery. In examining a placer, the same as a lode claim, the good faith of the claimant, and the value of the land for other purposes, are taken into consideration.

The Land Office records will show that only a small proportion of the cases on which contests have been initiated by the Forest Service ever come to a hearing, as the claimants do not file answers to the charges against the claims or do not put in an appearance before the Register and Receiver of the local Land Office at the date set for hearing. Final decisions by the Department of the Interior on all mineral cases contested since July 1, 1911, by the Forest Service in District 1, which comprises Montana, northern Idaho, eastern Washington, and western South Dakota, with the exception of one claim, have been favorable to the government.

The examination and protesting of worthless claims in the National Forests often protects the public from "wildcats." There have been instances, particularly in the outskirts of large districts, where companies have been formed and stock has been sold to develop ground without any real mineral value. To further the stock-selling schemes, patents have been desired and applied for, but when the claims were protested they have generally been abandoned.

Timber on a mining claim can be cut and used for the development of the claim. No permit from the Forest Service is necessary and no charge is made. The locator cannot, however, sell the timber or use it for any purpose foreign to the development of the claim. If more timber is needed for development than what is on the claim, it may be obtained elsewhere in the Forest without any charge under a free-use permit. When a mine has become productive and still needs more timber, a charge is made for it. The government, in making timber sales to outside consumers, always considers the local miners and leaves their supply unimpaired.

Had it not been for the establishment of the National Forests, a large part of the timbered areas now embraced within the Forests would by this time have passed into the hands of private interests as timber and stone claims. Once one of these claims is patented, the patentee not only controls the timber and surface rights, but any mineral that may be discovered on the ground. These non-mineral entries not only discourage prospecting, but tie up the mineral development of large tracts. In addition to being taken up under the non-mineral laws, these claims would gradually come into the possession of large timber companies

who have no interests in mining. Even an unpatented timber and stone claim is a drawback and discourages mining.

Any one who has ever had a mining claim conflicting with a claim of this kind can testify to the cost and annoyance of protesting the patent to the timber and stone claim and fighting protests against the issuance of patent on his own claim. The location of timber and stone claims cannot be made within the National Forests and any one is free to enter the Forests for prospecting and the location of legitimate mining claims.

The miners are gradually coming to realize that the reason for the difficulty in patenting some mining claims is in the law itself, and that the Forest Service has no arbitrary power to change the statutes.

Discussion

H. V. Winchell, Minneapolis, Minn.—There are two questions I should like to ask: First, does the Department still adhere to the decision made in the East Tintic case? Second, does the Department require placer ground to be of such value that, in the opinion of the Forest Service employees, it will pay to work, before admitting the location to be valid?

T. C. Hoyt,* Ogden, Utah.—With reference to the first question: the Department does not hold to that decision, as it was reconsidered by the Secretary of the Interior, and the Department is governed by the decision of the Secretary of the Interior, on those points always. The Department of Agriculture makes no decision as to what constitutes a sufficient discovery. The mineral examiners simply report their findings and opinions. These are transmitted to the chief of the field division of the General Land Office, who in turn transmits the reports to the Commissioner of the General Land Office, who decides whether the report raises an issue requiring a hearing. If a hearing is ordered, expert testimony is given on the question of the discovery, as well as the evidence of practical people who operate in the vicinity of the claim in question. As to the extent of placer values considered sufficient to justify a location, my answer will need a little amplification. I couldn't answer it directly for the reason that sometimes people resort to placer claim location as a means of acquiring land for other purposes than placer mining, which makes it absolutely necessary that the government give close attention to the question of whether good faith has been shown, as it is the evidences of good faith that largely determine whether a report adverse to a claim shall be made. That is a difficult point to cover sometimes. For instance, in the region I have recently visited there are some 17 miles of a valley, from $\frac{1}{4}$ to $\frac{3}{4}$ mile in width, all covered by placer locations. It is valuable stock-grazing land, and is stocked at the present time. It

*National Forest Service. Non-member.

also contains a stand of timber, and is on the watershed of an important stream—important from the fact that it furnishes the water supply to government storage reservoirs, and to pending reclamation and Carey Act projects. In this case the placer claims will not be disturbed as mere locations. They will simply remain there, and there will be no action with reference to them until the claimant asks for patent or, before patent, begins to denude that watershed and remove that timber. Then the government would direct that the mineral experts go on to the ground, examine it, prospect it, and submit their report, with a view to seeing if the claims were really valid and being used for actual mining purposes. Placer claims in the National Forests to-day are covering power sites, homesteads, dipping corrals, sawmills, summer resorts, and a good many other things, but a claim that is located as a *bona fide* placer claim and used for purposes consistent with mining will not be disturbed by the Forest Service.

This is an illustration of our attitude. In one of the National Forests where we had a withdrawal by the Secretary of the Interior for administrative purposes, the tract supported a cabin, pasture, and small field. Subsequently placer locations were made to cover this site and additional land up and down the canyon. The proposition was made to the locators, "If you will proceed with actual placer mining up to the point where you demonstrate to yourselves and to disinterested parties, competent to pass upon it, that you have a paying placer mine, we will get out of the way and allow you to mine all over the area, but we don't feel that we can do that on the showing as now given. Our experts have examined the land, and do not report sufficient values or sufficient prospects on your part to determine whether you have values enough to justify our giving up this land." This land was adjacent to a growing mining camp, and there were various reasons why we desired to know whether there were values there to justify before permitting it to pass to patent. These claims were not disturbed until the application for patent was made. Then it was a question of whether this 1½ or 2 miles should be alienated, or whether we should wait until values were proved. That case is now pending, and our proposition still stands. That is as near as I can illustrate the attitude of the Forest Service on that question.

MR. WINCHELL.—In the case of a quartz locator within the National Forest, where the report of the National Forest employees is to the effect that he has not made a valid discovery, what procedure is taken against him? Does he have further time and opportunity to perfect it, or is some action taken to oust him?

Mr. HOYT.—That answer will require a little amplification, too. Recently a mineral examiner of the Land Office submitted a report on a claim on the National Forest which was adverse. Feeling that more

expert testimony was desirable before deciding the issue, the examiner transmitted the request to me, and I detailed an examiner to examine this claim. Our examiner hunted up the locator, or present owner of the claim, and asked the claimant to accompany him on the examination, which was done. The owner of the claim said: "Now, I want to have your advice in this matter. If you think that the reports which you will have to submit, under the conditions as they now exist, are such as to jeopardize the probabilities of my getting a patent, I would like to have further time," and that proposition was accepted. The examiner transmitted that request with his report to me, and I transmitted the request to the chief of the field division, with the recommendation that no action be taken, and that the applicant be not required to defend his claim at the present time, but be given such time as he thought desirable in order to make a better showing, and mind you, this was on an application for patent. It should be borne in mind that these lands have been declared by the President of the United States, under authority of Congress, to be lands valuable for a certain purpose, a certain public purpose; therefore, before alienation is sanctioned, the Department having jurisdiction of the National Forest administration has to look carefully into these questions. The examination of mining claims in the National Forest by the Forest Service came about in 1906, when the Secretary of the Interior was petitioning for additional appropriations to examine lands. Congress did not appropriate to meet the requirements to the extent that the Secretary of the Interior thought desirable, so the President suggested at the time, "Can you not use the Forest officers throughout the National Forests in this respect?" and he directed the Secretary of Agriculture and the Secretary of the Interior to confer on this matter, and see if they could bring about some arrangement. The result was that the Secretary of the Interior instructed the Commissioner of the General Land Office to transmit instructions to the Register and Receiver, in all cases of application to patent lands in National Forests, to notify the Forest Service. The reports as to mining claims may in the first instance come merely from a ranger, who may not be qualified to pass on technical questions, but his report is placed in the hands of the mineral examiner, who is a man of considerable experience throughout the West, the one in my office at the present time being a graduate of the Columbia School of Mines. He examines the report, and if there are sufficient evidences of good faith, the claim is passed without question, the chief of Field Division being advised that there is no protest against the allowance of a patent in that case. If the evidence of good faith is manifest, we don't bother with technical examinations, or don't raise technical questions, as Mr. Gardner indicated in his paper; but if there seems to be a lack of compliance with law then the mineral examiner is directed to examine that claim, but he waives technicalities in every case where evidence of good faith is manifest.

The Commissioner acts upon the report, and no location of a mining claim in a National Forest is interfered with until he considers the report, or there is something in connection with it manifesting bad faith, failure to comply with the law, or interference with the administration. Sometimes we have had groups of claims placed right where examinations were being made with a view to sale of the timber, though the timber-sale contract had not been executed. In those cases we had the mineral examiner examine the claims carefully. He took very careful note of everything indicating the possibility of values and which threw any light upon the question of good or bad faith of the locators. In some cases we have had to go to the extent of asking for a hearing, with the result that the claims were canceled, and we proceeded with the sale of the timber.

D. W. BRUNTON, Denver, Colo.—In a case where two contiguous claims are held by the same owner, one of which has timber on it, and the other hasn't any timber, can a man operating a mine take the timber from the timber claim and use it on the other claim? I ask this because of a complaint that was made of a man who had two parallel claims in Idaho, both of which were being worked and producing, one claim being covered with timber, and the other absolutely bare. He claimed that the Forestry Department compelled him to purchase timber from them instead of taking it from the parallel contiguous claim which he owned, and which was covered with timber.

MR. HOYT.—The rule is that it is allowed without question; but it has happened in many instances that operators during the prospecting period of their mining have located surrounding ground with timber on it, and have made no effort whatever to develop that particular ground as a mine, but have simply used the timber off of it to develop a claim having actual mineral value. I can't understand why purchase was necessary in the case mentioned, because if the miner was simply prospecting he wouldn't have to purchase the timber, but if he has a paying mine then we would not allow him to locate timber land under mining claims without going ahead with *bona fide* development and showing mineral value on the timbered claim. In case the timber were required for a paying developed mine, they might require him to purchase at the rates prevailing in that immediate vicinity instead of allowing it to be taken from a located claim not showing mineral values.

The Evolution of the Round Table for the Treatment of Metalliferous Slimes

Discussion of the paper of Theodore Simons, *Trans.*, xlvi, 338.

HENRY LOUIS, Newcastle-upon-Tyne, England (communication to the Secretary*).—In this paper Mr. Simons derives the various forms of revolving slime tables, of which the Harz and the Linkenbach tables are typical examples, from the old rectangular or box buddle. This I believe to be an error, at any rate if Mr. Simons attaches the same meaning as I do to the "evolution" of a dressing appliance. By the "evolution" of such an appliance I understand the progressive development and improvement of an appliance acting on a definite principle, such improvements having usually for their object the economy of labor in working the same, or the increase of its capacity; when an entirely new principle is introduced, I no longer look upon this as evolution from the old type, but as the origination of an independent type of appliance. The box buddle was no doubt the predecessor of the old Cornish round buddle, which was evolved from it with the object of saving the labor required to work the box buddle; both machines are horizontal-current separators, in which the separation of sands of different densities is effected by their differential rates of fall in a horizontal current of water of considerable depth. These appliances, though well suited to sands, are for obvious reasons not adapted to the treatment of slimes. The revolving tables of the Harz and similar types, on the other hand, act on the principle which I have called separation in thin films of water, and which Mr. Simons calls the film sizing principle, in which the resistance of the surface over which the pulp flows plays an essential part in determining the separation of the particles, and which is well suited to the treatment of slimes. Unlike the last principle, it will not work in deep bodies of water and cannot be used where considerable depth of material accumulates in the appliance, as in the box buddle and round buddle. As I have shown in my book, *The Dressing of Minerals*, p. 326, these revolving tables are a development of the appliance known variously as the flat buddle, flat table, or frame, the last name, which is the old Cornish one, being the one that I personally prefer. These primitive appliances, as well as the continuous-acting round tables described by Mr. Simons (his Figs. 10 *et seq.*) all work on the principle of separation in thin films, and are all adapted for the treatment of slimes; his buddles and round tables (Figs. 1 to 9) do not work upon this principle, and should therefore be sharply differentiated from the former. The evolution in each case of the continuous acting, more or less automatic, circular appliance from the primitive, intermittent, hand-worked, rectangular appliance has proceeded upon strictly parallel lines. The question of the evolution of the respective types involves a point of some importance as regards the principles of ore dressing.

* Received Mar. 29, 1915.

Development of the Round Table at Great Falls

BY ARTHUR CROWFOOT, GREAT FALLS, MONT.

(Salt Lake Meeting, August, 1914)

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INTRODUCTION

The principal object of this paper is to present data on the development of the revolving convex round table as a concentrator for the through 0.07-mm. slimes from the ores of the Butte district, although some brief notes on the earliest recorded use of this type of concentrating table in the ore-dressing plants of the Lake Superior region and of the Butte district are also presented.

The revolving convex round table belongs to the class of concentrating tables known as film-sizing tables, these tables using the relative transporting power of a film, or thin sheet of water, as it flows over a quiet inclined surface, to separate the minerals of a sorted product.

The grains of heavy mineral move down the slope of the table deck slowly, or, in some cases, not at all, after the initial force of the feed entry

is expended, since, being smaller, they are acted upon only by the slowly moving portion of the water which is in contact with the surface of the table; on the other hand the grains of light mineral (gangue), being larger, are exposed to the more rapidly moving water of the upper current and are therefore moved much faster down the slope of the table.

The speed with which the coarser grains of gangue material move down the slope of the table is also accelerated intermittently by the action of the waves of pulp flowing down the table. The coarser grains of gangue material are submerged, partly in the slow-moving under current and partly in the swifter upper current, and their speed of travel is greater than that of the former and less than that of the latter. Observations made on the action of the pulp on the table have shown that these coarse grains of gangue material are subject to distinct blows from the waves of pulp which momentarily increase the speed of these grains.

The speed at which the under current, or friction film, moves is dependent upon the character of the surface of the table, a rough surface retarding the flow of this current while a smooth surface accelerates the flow.

The revolving round table is a development of the stationary Cornish buddle, one of the oldest forms of slime concentrators, and it is a fact of great interest that a machine so closely related to one of the oldest forms of concentrating machines should be selected as the right machine for an important position in the flow sheet of our latest milling practice on copper sulphide ores in Montana.

The origin of the buddle seems to be lost in obscurity, and there is considerable doubt as to whether Cornwall or Germany first introduced this method of concentration. Cornwall has laid claims to having taught the art of tin mining and metallurgy to the Germans, and a story is related by one Matthew Paris, a Benedictine monk, by birth an Englishman, who died in 1259, that a Cornishman who fled to Germany on account of a murder which he had committed, first discovered tin there in 1241, and that in consequence the price of tin fell greatly. This would seem to point to the probability of the buddle having originated in Cornwall.

Georgius Agricola in his *De Re Metallica* devotes considerable space to descriptions of dressing implements which were similar to the early forms of the box buddle. These consisted merely of a wide inclined stationary trough, with twigs or riffles placed so as to catch the heavy particles, the principle being the same as that used in Cornwall to-day. Agricola was born in 1494, and therefore this method of dressing or concentrating ores evidently dates back earlier than this time.

In the third century B. C. the Greeks practiced a system of milling and concentrating, using a method of washing the crushed ore which was very similar in some respects to the box buddle.

Edward Ardaillon in *Les Mines du Laurion dans l'Antiquite* gives the following details:

"From the mills it (crushed ore) was taken to washing plants, which consisted essentially of an inclined area, below which a canal, sometimes with riffles, led through a series of basins, ultimately returning the water again to near the head of the area. In washing, a workman brushed upward the pulp placed on the inclined upper portion of the area, thus concentrating there a considerable proportion of the galena; what escaped had an opportunity to settle in the sequence of basins somewhat on the order of the buddle."

The square or box buddle of Agricola's time would hold about 600 lb. of material when full. Owing to the necessity of returning the lower two-thirds of the material deposited to be re-treated on the same buddle, also to the amount of labor required to operate it, this form of buddle is practically obsolète everywhere.

The round buddle works on the same principle as the box buddle, and could be described as a series of box buddles arranged radially, with their feed ends in the center.

To obviate channeling of the ore, arms are attached to the rotating feed gear, and on these arms are hung strips of brattice or coarse cloth, which gently drag over the deposit, and thereby keep the surface even. This type of buddle is in use both in the concave and the convex forms. In Cornwall it is very extensively used in tin dressing, also in Derbyshire, Isle of Man, and Cumberland for concentrating galena ore.

The Cornish buddle or building table, which was used almost exclusively as a concentrator for tin ores in Cornwall during the early days of tin mining in that district, was usually from 18 to 20 ft. in diameter and was built in the form of a circular tank with sides 18 in. deep, the bottom sloping gently from the center to the circumference. In the center of the tank or table was a mound with a conical top, the mound being from 5 to 6 ft. in diameter. Pivoted in the center of the mound was a vertical shaft which revolved. This shaft carried a pulp distributor and four wooden arms which extended out over the buddle, the arms being equipped with brushes made of cocoa matting frayed at the ends or wisps of brush which dragged lightly over the sand as it accumulated in the buddle. These brushes kept the surface of the charge trued up and hence assisted the work of concentration very materially by preventing channeling.

During the process of running a charge in a buddle the water and slime were drawn off at points on the circumference which were equipped with sliding doors perforated with auger holes. As soon as the buddle became filled, the foreman examined the contents, marking off with the point of his shovel four or five concentric circles on the surface of the charge, their relative position depending on the richness of the mineral within each.

The buddle crew then shoveled the outer ring, which was mostly slime, into a launder which delivered it to a settling pond; the next ring was thrown into a waste tailing launder, the intermediate rings were shoveled into wheelbarrows and carried to other buddles for re-treatment, while the inner ring of all, containing about 25 per cent. black tin, was sent to the calciners.

The total result of this first buddling was that about one-quarter of the waste was removed from the pulp treated; another small part was sufficiently enriched to be sent to the calciners, while probably more than one-half of the whole had to be re-treated.

This method of buddle treatment has been supplanted in the more progressive plants by the use of Frue vanners and revolving round tables.

American milling practice has always been averse to the use of concentrating machines which operate intermittently, or machines which produce a large amount of middling material for re-treatment, therefore, the revolving round table, which at once overcame so many of the difficulties of the operation of the stationary buddles, made rapid headway in American mills in displacing stationary buddles, kieves, and other intermittent machines.

The Round Table in Lake Superior Practice

The milling practice in the Lake Superior district has been described in papers read before the Institute by Charles W. Rolker,[1] H. S. Munroe,[2] and F. G. Coggin.[3] As nearly as the writer can learn the first revolving round table was introduced in the Lake Superior region in 1873, being installed in the Atlantic mill by a Mr. Jenkins, a Cornishman who had had considerable experience in the tin-ore dressing plants of Cornwall; and by 1883 it was in general use in the district. The round table most largely used was the design of W. J. Evans, the distinctive feature of which was the "dead-head" at the center, from 6 to 8 ft. in diameter, one-half of which was used for distributing the slime and the other half for distributing clear water.

The part played by the revolving round table in the flow sheet of the Michigan mills was to treat the spigot discharges of the settling tanks receiving fine product from the stamp mills, the material coming to the tanks as overflow material of the hydraulic classifier.

Before the advent of the reciprocating table, the round tables were used to produce a finished concentrate; two decks on a shaft were generally used, the middling from the upper deck being re-treated on the lower deck or on a separate system of tables.

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[1] *Trans.*, v, 584 to 606 (1876–77).

[2] *Trans.*, viii, 409 to 451 (1879–80).

[3] *Trans.*, xii, 64 to 68 (1883–84).

The writer is informed by C. H. Benedict of the Calumet & Hecla company that the present practice at its mills is in no case to make a finished concentrate on the round tables, these tables being always followed by Wilfley tables, either two, three, or even six decks of the round tables being dressed up on one Wilfley table.

Round tables are in use at the C. & H., Tamarack, Ahmeek, Osceola, Isle Royal, and Mass mills. Ordinarily there are three decks on one shaft except at the C. & H. mills, where there are four decks on a shaft. The tables make 1 rev. per minute, and in new installations cement deck surfaces are used to replace the wood deck surfaces of the older installations.

The slope of deck surface in general is 1.5 in. per foot although a slope of 1.25 in. per foot is sometimes used on fine material. The tonnage fed per deck ranges from 15 to 20 tons per 24 turns and the recovery made is ordinarily from 40 to 50 per cent. On low-grade ores the ratio of concentration may reach as high a figure as 80 into one and is never less than 30 into one. With classifiers now in use the maximum size of grain going to the round table is about 0.25 mm.

The revolving round table used at the present time by the Calumet & Hecla Mining Co. consists of four decks mounted on a shaft. The table is driven from the top by means of a crown gear and pinion and is timed to make 1 rev. per minute. The total height of the table from crown gear to floor line is 21 ft. and the spacing between the decks is 3 ft. 8 in. The decks have the usual umbrella frame, the supporting braces consisting of 2-in. angle irons. The frame is so designed that the supporting braces occupy very little of the space between the decks. The circumference of the decks is stiffened by 3-in. angle-iron rim segments bent to the proper curve. The deck surface used is cement laid on wood, the cement being about ½ in. thick.

Each deck is equipped in the center with a cone with a 45° slope and a base diameter of 3 ft. The feed pulp is delivered on one side of the table only from a special form of feed box which delivers the pulp against the face of the 45° cone. The decks are in general given a slope of 1.5 in. in 1 ft.

Early Use of Revolving Round Table in Butte District

From C. W. Goodale's paper on the Concentration of Ores in the Butte District, Montana,[4] it would appear that the Colorado Smelting & Mining Co. was the first to install round tables in the Butte district. The plant started operations in 1882. Round tables were installed in the initial flow sheet of this mill, but were afterward discarded in favor of Frue vanners.

[4] *Trans.*, xxvi, 602 (1896).

In the concentrator of the Montana Copper Co. the reverse is the case, as Frue vanners were first installed and later were replaced by round tables.

Use of the Round Table at Great Falls for Sand-Slime Feed

Construction work on the first concentrator of the B. & M. C. C. & S. Mining Co. at Great Falls was started in the spring of 1891, the concentrator being put into operation in March, 1892. A second concentrator was completed and put into operation in the fall of 1900. Convex revolving round tables were installed for the treatment of the fine sand and slime which overflowed the Evans hydraulic classifiers.

The round-table practice at Great Falls, preceding the development of the table as a concentrator for through 0.07-mm. slime, did not differ greatly from the round-table practice in other Montana mills; usually the feed ranged in size from 50 mesh (0.36 mm.) to zero, with from 3 to 5 per cent. of coarser material due to the imperfections of the hydraulic classifiers employed, or to the overloading of these classifiers. No very close classification of feed was attempted, nor was any attempt made to produce a waste tailing from the round tables, the tables being guarded by the vanners. The use of the Wilfley table, under the conditions of round-table operation, rapidly superseded the use of the round table for the following reasons: (*a*) A cleaner grade of concentrate was produced which contained as high or higher a percentage of the copper fed to the table; (*b*) a large tonnage of fairly low-grade tailing could be produced from a feed no more closely classified than the feed sent to the round tables, thus eliminating the re-treatment of this material on vanners; (*c*) the Wilfley table acted as a classifier in that it made a separation between the sand and slime fed to it.

It is probable that good work could be obtained from round tables when fed with sand ranging in size from 60 mesh (0.25 mm.) to 200 mesh (0.07 mm.). Under these feed conditions, the very fine slime being eliminated from the feed, the tables should produce a large amount of low-grade tailing product.

The Round Table for Fine Slime

The use of the revolving convex round table as a concentrator for a through 0.07-mm. slime feed was originated by J. M. Callow of the General Engineering Co., Salt Lake City, in an exhaustive series of tests made by him on the concentration of overflow slime at the Boston & Montana concentrator, Great Falls, during the period October to December, inclusive, 1904.

This is believed to be the first time that so fine a feed had been treated alone on a round table; the writer has looked up all available

literature on the subject but can find no record of a feed which did not contain a large percentage of coarser sand, the feeds usually varying from 0.35 mm. down.

The concentration of the slime was preceded by a dewatering or pulp-thickening operation, the slime as produced in the mill being unwatered in a peripheral overflow tank of Mr. Callow's design, since well known under the name of the Callow settling tank or cone.

By the use of this unwatering device, preceding the concentrator, the density of the slime was increased from about 70 g. of solids per gallon of pulp (2 per cent. solids) to about 350 g. of solids per gallon of pulp (8.75 per cent. solids) with a loss of only from 2.0 to 2.5 per cent. of solids and copper in the overflow of the tank. The capacity per Callow tank, to obtain the above results on B. & M. slime, was 25 gal. of slime feed per minute or 36,000 gal. per 24 hr. per tank, the tank being equipped with a goose-neck spigot discharge with an opening $\frac{3}{8}$ in. in diameter operating under a head of 24 in. A full description of Mr. Callow's tests will not be attempted in this paper, the general results only being given.

The machines at Mr. Callow's disposal for trying out the concentration of the thickened slime pulp from his tanks consisted of a smooth-belt Frue vanner, a Wilfley table, and a revolving convex round table.

The round table was not at first considered by Mr. Callow, the first four tests being made on a Frue vanner and the next 16 on a Wilfley table. The copper recoveries made by the Frue vanner and the Wilfley table were in both cases low, especially so in the case of the Frue vanner, which was not equipped with the necessary stroke adjustments for the treatment of so fine a feed, which it must be remembered was about 96 per cent. finer than 0.07 mm. (200 mesh). Mr. Callow then directed his efforts toward finding what could be accomplished by using the revolving round table as a concentrator for the slime and his first results were so satisfactory that the remainder of his tests were made on this table.

No mechanical adjustments were possible except the speed and this remained unchanged from the speed used for the regular sand-slime feed, 1 rev. in 2 min. The Wilfley table and vanner have been developed as slime concentrators since this period but the development of the round table has enabled it to maintain its supremacy in this particular branch of ore dressing.

The tests made by Mr. Callow included tests made on the individual slimes and on various combination of the slimes, the end results of which showed that the Callow tanks would handle from 25 to 30 gal. of slime pulp per minute, thickening this slimy water from a density of about 60 g. of solids per gallon of pulp to a density of about 350 g. (1.5 to 8.75 per cent. solids), overflowing about 90 per cent. of the water contained in the slime and delivering from 95 to 98 per cent. of the solids fed in the pigot discharge for treatment on the round tables.

The revolving convex round table was shown to have a capacity for treating from 4 to 5 tons of the thickened slime per 24 hr., producing from ¾ to 1 ton of concentrate assaying from 5 to 6 per cent. copper and 71 to 75 per cent. insoluble, and containing about 50 per cent. of the copper in the feed, the ratio of concentration being from 5.5 to 6 tons into one.

It was recommended by Prof. R. H. Richards that an Embrey vanner be tried out as a concentrator for the thickened slime pulp from the Callow tanks, as, in his opinion, the end-shake vanner, to which type the Embrey vanner belongs, would prove to be a superior machine to the side-shake vanner for the concentration of the fine slime.

An Embrey vanner was, therefore, installed in the concentrator and thoroughly tested on the treatment of the slime during March and April, 1905. The results of the tests confirmed Professor Richards's statement in regard to the fact that the end-shake vanner would develop a higher efficiency than the side-shake vanner in the treatment of very fine slime, the recovery of copper being increased, but the recovery made by the Embrey vanner still fell short of the recovery made by the revolving convex round table and the capacity of the vanner was much lower. The main data received from tests made on the Embrey vanner follow (p. 425).

Some results obtained from tests made in the Frue vanner when equipped with a corrugated rubber belt are also presented.

Construction of Slime Plant at Great Falls

In order to try out Mr. Callow's flow sheet on a commercial scale it was decided to erect a slime plant with a capacity for treating about 1,000,000 gal. of slime per 24 hr., the pulp carrying about 80 tons of solids. This required an installation of 30 Callow tanks and four four-deck round tables. Construction was started on this plant during the summer of 1905 and the operation of the plant was started on Oct. 1, 1905.

The building was so designed, and the equipment so arranged, that the plant could be readily enlarged to the full capacity required should the results obtained justify this course.

The original dewatering and concentrating equipment of the slime plant consisted of 30 8-ft. Callow tanks and 16 17-ft. revolving round tables arranged as four four-deck tables. The round tables used were the two-deck tables removed from the concentrator at about this time, two double decks being set up one above the other and the shafts connected with a coupling so as to form a four-deck table.

Of the 16 decks installed, four were given a linoleum cover, four a cement surface laid on the original wood surface, and on the remaining eight the original wooden surface was maintained. The circular feed-distributing apron with a 2 in. to the foot slope, previously mentioned,

was allowed to remain on the tables. The slope of the deck surfaces was 1.25 in. per foot for the wood and linoleum surface decks and 1.5 in. per foot for the cement surface decks; the speed of all tables was 1 rev. in 100 sec. or 36 rev. per hour. The feed pulp was distributed over one-half of the table and washed with dressing water on the other half of the table, the tables being operated in practically the same manner as when treating a sand-slime feed in the concentrator.

The cement-surface decks installed in the slime plant were the first decks of this character installed at the Great Falls works; cement-surface decks were coming into general use in the Michigan ore-dressing plants at this time and probably had been used at other plants; in fact, John A. Church mentions having installed cement-deck round tables at Tombstone, Ariz., in 1883.[5]

Mr. Church states that the use of cement for round-table deck sur-

Results of Tests Made on the Embrey Vanner

Date 1905	Data on Feed to Vanner: No. of Tests Made	Rate per 24 Hr.: Gallons	Rate per 24 Hr.: Pounds	Density in Grams per Gal.	Assay Per cent. Cu	Concentrate: Assay Cu	Concentrate: Per cent. Insol.	Assay Per cent. Cu in Tailing	Ratio of Concn. Tons to 1	Recovery of Cu Per cent.
Group No. 1										
Rev. per min. 225, 1-in. stroke, belt speed 6 ft. 8 in. per min., slope 4 in. in 12 ft.										
Mch. 8	1	3,850	5,100	600	2.4	9.5	64.0	1.7	11.7	33.0
Group No. 2										
Rev. per min. 225, 1-in. stroke, belt speed 9 ft. 6 in. per min., slope 6 in. in 12 ft.										
Mch. 9–14	6	4,210	3,400	364	2.7	6.8	73.0	2.13	8.3	30.5
Group No. 3										
Rev. per min. 225, 1-in. stroke, belt speed 10 ft. per min., slope 9½ in. in 12 ft.										
Feb. 23–28	6	3,730	3,930	473	3.2	10.0	60.0	2.53	9.75	32.5
Group No. 4										
Rev. per min. 225, 1-in. stroke, belt speed 12 ft. 9 in. per min., slope 11½ in. in 12 ft.										
Mch. 1–7	5	4,050	3,900	439	2.48	6.95	71.0	1.91	7.63	36.8
Group No. 5										
Rev. per min. 225, 1-in. stroke, belt speed 16 ft. per min., slope 18 in. in 12 ft.										
Mch. 7–25	28	4,286	3,818	394	2.46	7.54	69.0	1.99	7.21	37.3
Group No. 6										
Rev. per min. 248, ¾-in. stroke, belt speed 16 ft. per min., slope 18 in. in 12 ft.										
Apl. 14–22	11	5,829	5,199	425	2.3	7.20	68.0	1.7	9.4	37.8
Group No. 7										
Rev. per min. 247, ¾-in. stroke, belt speed 18 ft. per min., slope 18 in. in 12 ft.										
Apl. 24–26	3	5,887	5,416	495	2.5	7.70	68.8	1.8	10.8	31.0

[5] *Trans.*, xv, 601 (1886–87).

A reference to the foregoing table shows that the wood deck surface recovered 6.86 per cent. more of the feed copper in concentrate than the linoleum-covered deck and 8.33 per cent. more than the cement deck surface, but also, that this additional recovery was made at the expense of a more siliceous concentrate and a concentrate with a lower copper value, the wood-deck concentrate assaying 0.63 per cent. less in copper than the linoleum-deck concentrate and 1.74 per cent. less than the cement-deck concentrate. From the above it follows that the wood deck produced a lower tonnage of a lower grade of tailing than the other decks and this is shown to be the case, the wood deck producing 0.358 ton less tailing than the linoleum deck and this tailing assaying 0.10 per cent. less in copper, and 0.597 ton less tailing than the cement deck assaying 0.06 per cent. less in copper.

It may be well to parallel additional data on the tailing losses from the tables and this is done in the following table:

Data on Tailing Losses

	Kind of Deck Surface		
	Wood	Linoleum	Cement
Per cent. of feed solids in feed-side tailing	31.75	36.11	36.39
Per cent. of feed copper in feed-side tailing	20.16	24.39	24.10
Assay per cent. copper of feed-side tailing	1.77	1.86	1.82
Per cent. of feed solids lost in wash-side tailing	35.84	38.20	42.49
Per cent. of feed copper lost in wash-side tailing	17.70	20.33	22.09
Assay per cent. copper of wash-side tailing	1.35	1.45	1.44
Per cent. of feed solids lost in total tailing	67.59	74.31	78.88
Per cent. of feed copper lost in total tailing	37.86	44.72	46.19
Assay per cent. copper of total tailing	1.55	1.65	1.61

The tailing made by the linoleum and cement decks does not show a notably higher assay copper percentage than the tailing produced by the wood deck, especially so in the case of the cement deck, the greater loss of copper in tailing being largely due to an increased production of tailing material. For this reason the writer favored the use of cement deck surfaces with a slightly flatter slope than the deck under test (this deck having a slope of 1.5 in. per foot), providing the cement was laid on a rigid framework so that the cement could not crack or get out of true in any way.

Low-Tonnage Test Results.—The reduction of the tonnage fed to the decks per 24 hr. from 4.5 to 3.5 tons, improved the copper recovery and reduced the tailing loss on all three decks but had the greatest effect on the work of the linoleum and cement decks, the recovery made by the latter decks approaching to within 3.0 per cent. and 1.4 per cent. respectively of the recovery made by the wood deck, this being partly due to a slight lowering of the grade of concentrate produced in each case.

A comparison of the results obtained in the low-tonnage tests showed that the additional loss of copper in tailing from the linoleum and cement decks was due entirely to the production of a greater tonnage of tailing material than was produced by the wood deck, the linoleum deck discarding 3.64 per cent. more of the feed solids as tailing and losing 2.99 per cent. more of the feed copper than the wood deck, and the cement deck discarding 5.10 per cent. more of the feed solids as tailing and losing only 1.37 per cent. more of the feed copper. These results also favored the use of cement deck surfaces as tending to produce a cleaner grade of concentrate and a lower grade of tailing with a lower consumption of dressing water if rigidly constructed and given the right slope. During the running of above tests the writer noted the fact that no concentration took place on the feed aprons previously described, these aprons having a slope of 2 in. in 1 ft., and also noted that very little settlement took place on the first 8 or 10 in. of the working radius of the table, due to the velocity that the steep slope of the apron and the fall of about 1 in. from surface of apron to surface of table deck gave to the feed pulp.

Central Feed Apron Removed

At the conclusion of the tests, February, 1906, the writer removed the feed apron from one of the wood decks, filled in the space with a cement mortar (3 sand, 1 cement), with a finishing coat of neat cement. A slope of but 1 in. in 1 ft. was given to this surface since it was considered that the initial velocity of the feed when discharged from the feed box was sufficient to warrant lesser slope than that given to the remainder of the deck surface.[6] The results of this change showed that a considerable deposit of very clean concentrate could be obtained within 12 in. of the edge of the wash-water cup, the working radius of the table being lengthened about 3 ft. in the direction of the center. Shortly afterward another of the wood decks was changed in the same manner and a new type of feed and wash-water cup was introduced with a view to getting a better delivery of feed on the table. These feed and wash-water cups, which for experimental purposes were made of galvanized iron, were so constructed that the feed and wash water overflowed the rims, falling down the sides on to an apron with 3-in. radius and 15° slope, the outer edge of the apron practically touching the deck surface. These feed and wash water cups, if kept perfectly level and given attention by the operator about twice per shift to remove any foreign material, gave satisfactory service, delivering the feed and wash water to the table with an even distribution.

Overflow feed and wash-water cups had been previously used on the round tables, but on these the apron attached to the cup had a slope of

[6] This is the first time that a variable deck slope was tried at Great Falls.

45° and the edge of the apron was usually from 6 to 8 in. above the surface of the distributing apron on the table. On all cement deck surfaces constructed since this time the central distributing apron has been dispensed with, the working radius of the table being extended to the feed and wash-water cup.

Re-treatment of Round-Table Tailing

In June, 1907, the writer carried out a series of tests to determine to what extent the recovery of copper could be increased by re-treating the round-table tailing.

The feed and wash side tailings from six round tables (four wood and two linoleum deck surfaces) were fed into four Callow tanks, the plug discharges of the tanks in turn being fed on to a cement-deck round table, while the overflow went to the tail race. The cement-deck table made a low recovery of copper (21.29 per cent.) which when figured back to original slime gave an additional copper recovery of about 2.5 per cent. The concentrate produced was of a very low grade, being not much richer than the original slime.

A test was then made on the re-treatment of the feed-side tailing alone. A better grade of concentrate was obtained than from re-treating the mixture of feed and wash side tailings, but the recovery made was lower. The tests showed conclusively that the amount of original slime copper to be recovered by a re-treatment of the round-table tailing was small, probably no more than would be gained by a first treatment with tables properly designed for the treatment of a slime feed.

Determination of Conditions for Round-Table Treatment of Slimes

Early in 1908 the writer had an experimental slime table constructed similar to the one described in Richards's *Ore Dressing*, vol. ii, p. 1155, with a view to carrying out a series of experiments to determine the proper slope of deck surface, and rate of feed, for round tables operating on a through 0.07-mm. feed. The table, which was 1 ft. wide and 6 ft. long, was made adjustable as to slope by an arrangement of steps which increased the slope by $\frac{1}{8}$ in. per foot at one time. A smooth cement surface was used on the table and the various slimes and slime mixtures were tested out under varying rates of feed, deck slopes, etc.

The conclusions reached from the results of the tests were as follows: The table should be 18 ft. in diameter; should have a cement deck surface sloping 1.25 in. in 1 ft.; and should make 1 rev. in 200 sec. A re-treatment of the feed-side tailings on a canvas deck surface was recommended.

During the summer of 1909 we carried out a series of tests in the concentrator, using an adjustable segment of a canvas-covered round

table. The purpose was to determine the difference in behavior of the various concentrator slimes, as to both settling and concentration, and the proper treatment for each.

The experimental table as originally constructed was a segment of a round table, being the equivalent of one-tenth of an 18-ft. table. The slope of the table was adjusted by raising or lowering the feed end. The surface used on the table deck was No. 6 duck, the woof (cross threads) being laid approximately parallel to the direction of the slope. A large number of tests were made, both on the individual slimes and upon certain slime mixtures such as the slime plant was handling at the time.

The following general conclusions were arrived at: The primary slimes might be mixed, dewatered so as to recover about 98 per cent. of the solids, and then treated on a canvas-deck round table with a slope of 1.25 in. per foot. The table should make 1 rev. in 20 min. and produce a concentrate assaying 7.0 per cent. copper, and a tailing assaying 1.15 per cent. copper. For the lower-grade or secondary slime a slightly greater deck slope (1.375 in. per foot) was suggested, but it was considered probable that the 1.25 in. per foot slope would be found quite satisfactory for the normal mixture.

Toward the latter part of 1907, Dr. Robert H. Richards suggested the use of canvas-deck round tables for the slime plant, the tables to be given a speed of 1 rev. per hour and to be fed over nine-tenths of their entire surface, no dressing water to be used. Acting on this suggestion three round-table decks were covered with canvas (No. 6 duck) and the speed was reduced to 1 rev. per hour. It was expected that the concentrate would be so low in grade as to require a final dressing on another table, but such was not the case, as a very fair grade was produced.

The slime plant practice recommended by Dr. Richards was as follows: A low ratio of concentration, a low-grade concentrate and a high recovery on the round tables; the concentrate to be re-treated and enriched on a secondary table, from which the tailing would go back to the round table. In accordance with this suggestion, the round tables were recommended for use as roughing tables. Dr. Richards also recommended that the Embrey vanner be used as a finishing machine for cleaning up the rough concentrate from the round tables, but up to the present time this part of the plan has not been tried out, Deister, Craven, and James tables having been used as finishing machines.

Variation in Slope of Round-Table Deck

The first round table with a variation in the slope of the deck surface, tried out at Great Falls, with the exception of the change made when removing the central feed aprons, was installed in the slime plant in June, 1910.

J. H. Klepinger, Assistant Superintendent, suggested the use of a conoidal deck surface, and proposed to lay a cement surface of this character on one of the wooden decks. As the wooden decks in use were old tables, and as the cement deck would add greatly to the weight which the framework would have to support, it was thought that the structure would break down under the increased load. A compromise was therefore adopted, a cement deck surface being installed which was divided into two sections, at first of the same length radially, the feed apron previously referred to being left in the center, but of unequal areas, the upper deck being concentric to the lower deck. There was a drop of about 1.25 in. between the first and second deck sections. The first or upper deck section was given a slope of 1 in. in 1 ft. and the second or lower deck section a slope of 1.25 in. per foot. The steep feed apron with a slope of 2 in. per foot was afterward removed and the upper deck with its slope of 1 in. per foot was extended to the feed box.

This change gave the upper deck a radial length from the point where it received the feed, to its perimeter, of 5 ft. About 3 in. of radial length was occupied by the slope of the upper to the lower deck, leaving a radial length of 30 in. for the lower deck.

Data From Five Tests on Cement Step Deck

	Date of Test Period				
	June, 1910	Sept., 1910	April, 1911	March, 1913 No. 1	March, 1913 No. 2
Time for table to make 1 rev., min.....	60	19	19	19	19
Tons dry solids treated per 24 hr.....	5.84	11.3	5.4	6.184	48.25
Density of feed in grams per gallon...	314	432	364	495	246
Assay per cent. copper in feed.....	2.44	2.21	3.70	3.24	3.16
Tons total concentrate produced per 24 hr.....	0.750	3.2	2.2	2.558	1.220
Assay per cent. copper in concentrate.	7.75	4.83	7.19	5.98	8.64
Assay per cent. insoluble in concentrate	70.8	75.8	70.8	71.5	57.9
Assay per cent. FeO in concentrate....	8.9	7.5	9.1	9.9	16.1
Per cent. of copper fed, recov'd in conct.	31.93	61.15	79.45	76.31	68.52
Ratio of concentration, tons into one..	7.750	3.6	2.5	2.38	3.95
Ratio of enrichment (copper).....	3.17	2.18	1.94	1.84	2.73
Tons feed-side tailing prod. per 24 hr..	2.51	4.02	0.9	1.388	2.044
Assay per cent. copper.....	1.58	1.08	1.03	1.58	1.22
Tons wash-side tailing prod. per 24 hr..	2.58	4.14	2.3	2.238	1.572
Assay per cent. copper.....	2.24	1.30	1.5	1.14	1.46
Tons total tailing produced per 24 hr..	5.1	8.2	3.2	3.626	3.616
Assay per cent. copper.....	1.9	1.19	1.28	1.31	1.33
Gallons of fresh water used per 24 hr...	8,382	12,347	9,417	15,659	16,783
Gallons of fresh water used per ton fed.	1,435	1,093	1,744	2,530	3,480
Per cent. of original feed solids in conct.	12.91	28.04	40.85	41.36	25.07
Per cent. of original feed solids in tailing.	87.09	71.96	59.15	58.64	74.93

This table was tested in June, 1910, being set to make 1 rev. per hour. The results obtained were not satisfactory, a heavy building up of solids occurring on the upper deck which caused a channeling of the feed and consequent heavy tailing loss. The speed of the table was then increased

so that it made 1 rev. in 19 min. or practically 3 rev. per hour, the table being again tested in September, 1910, and April, 1911. In spite of the fact that the tonnage fed to the table in the September test was nearly double the tonnage fed in the June test, the work of the table showed great improvement, the channeling of the feed being nearly entirely eliminated. The April test was made on a lower tonnage, good results being obtained. The table was again tested in March, 1913, while the writer was carrying out some tests on conoidal deck tables, the steep feed apron having been removed in the meantime, two tests being made under different tonnage and feed-density conditions. The test results showed that the removal of the steep feed apron further improved the work of the table.

In the two 1910 tests and the April, 1911, test the table was operated as a roughing table, while in the March, 1913, tests the table was fed over about three-quarters of its area and washed with dressing water over about one-quarter of its area; this accounts for the smaller quantity of fresh water used in former tests.

In the April, 1911, test the central feed apron had been trued up so that a better distribution of feed was obtained.

In comparing the results of the two 1910 tests and the April, 1911, test it can be seen that the work of the table improved greatly with each succeeding test. In the June, 1910, test the slow speed of the table caused channeling even with a light feed. In the September, 1910, test the speed of the table and the tonnage were both increased, and although the tonnage was increased rather too much, the work done showed considerable improvement. In the June, 1910, test and the April, 1911, test the tonnages treated by the table were very nearly the same so that the effect of the increased speed can be better observed in these comparisons. The assay per cent. of copper of the feed in the April test was higher than in the June test, but the tailing produced was lower and the work done by the table showed great improvement. It was considered doubtful at the time, however, whether the work by this type of table would surpass the work done by a rigidly constructed cement-deck table with a regular slope of 1.25 in. per foot and so far these doubts have been justified.

The Steel-Frame Cement-Deck Round Table

The construction of a four-deck steel-frame cement-deck round table (Fig. 1) was started early in 1911, the table being installed in the slime plant and put into commission on Apr. 17, 1911.

Of the four decks, the three upper decks were given a slightly rough finished cement surface while the lower deck was cemented in such a manner that canvas could be laid over the cement, the canvas being tacked to strips of wood laid radially in the cement. By this arrangement

it was possible to change the deck readily from a canvas to a cement surface if the canvas proved unsatisfactory.

When the steel-frame table was designed it had already been proved that the rough canvas surface was satisfactory for a roughing table designed to produce, with high efficiency, a low-grade tailing and a low-grade concentrate. The disadvantages of the use of canvas were, the cost of maintenance and excessive amount of wash water required.

It was evident that if a cement surface of a character similar to canvas could be utilized, these disadvantages could be eliminated. It was for this reason that three of the decks of the new table were surfaced with cement at the start. The one canvas surface was soon replaced with cement.

The steel frame was a great improvement over the wood-frame construction; the cement decks being rigidly supported, a true conical surface could be maintained and the liability of cracks developing in the deck surfaces was eliminated. The cement decks were made of a mixture of one part cement to two parts concentrator-tailing sand.

The decks had a diameter of 18 ft. and a slope of 1.25 in. per foot, the distance between deck surfaces being 5 ft. The total height from floor to top of center shaft is 25 ft. 2 in. The diameter of the center shaft is 8 in., the shaft consisting of five cast-iron sections, a top and a bottom section and three intermediate sections, the intermediate sections being 5 ft. 1½ in. long.

The tables were equipped with movable sprays for washing off the concentrate. These were supposed to wash cleaner with less consumption of water than stationary sprays. The sprays had a throw of 9 in. transmitted by means of an eccentric and crank and made 4 strokes per minute; the use of movable sprays was afterward discontinued, since no especial advantage was shown over the stationary form.

Tests on Cement and Canvas Deck Surfaces

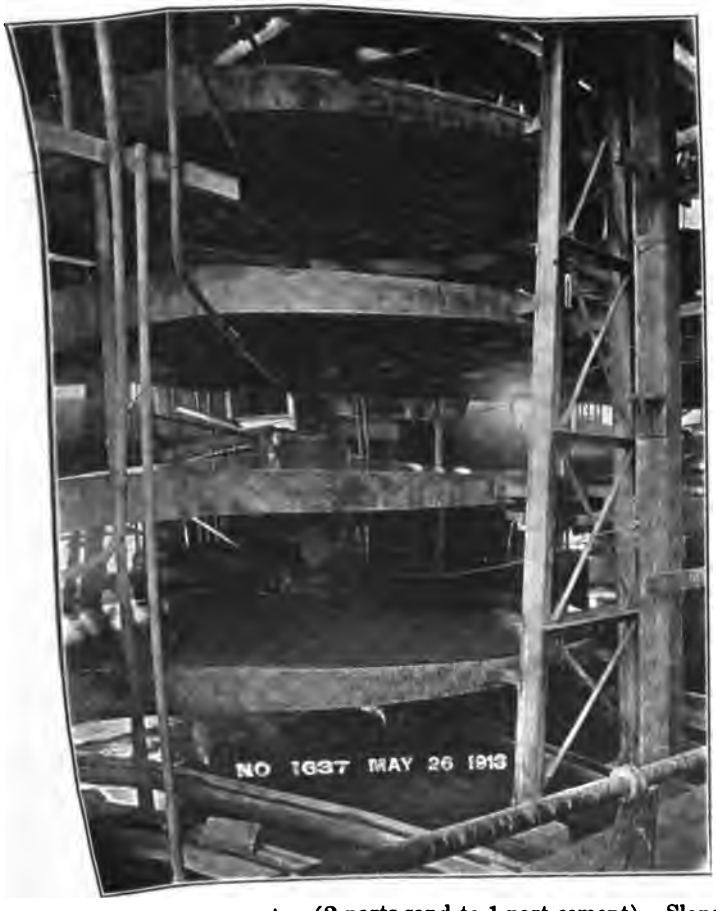
One cement deck surface and the canvas-covered deck were thoroughly tested in April, 1911, the results of the tests being presented below.

It has been stated above that the decks had a slope of 1.25 in. per foot, but it was found from accurate measurements made on April 29, 1911, that the actual slope of the cement deck tested was 1.2 in. per foot and of the canvas deck 1.13 in. per foot. The chief data obtained from the tests are placed in parallel columns.

In the following tests the two tables were fed at the same rate per 24 hr., the feed for the canvas deck, however, being slightly higher in assay per cent. of copper than the feed for the cement deck.

The cement deck made a recovery of 82 per cent. of the copper fed in

2.9 tons of concentrate, the ratio of concentration being 2.9 tons into one. The canvas deck made a recovery of 87 per cent. of the copper fed in 4.3 tons of concentrate, the ratio of concentration being 1.9 tons into one. If these concentrates were sent direct to the smelter in the condition



Deck surface, cement mortar (2 parts sand to 1 part cement). Slope of deck surface, 1.25 in. per foot. Diameter of decks, 18 ft. Decks make one revolution in 20 min.

FIG. 1.—FOUR-DECK STEEL-FRAME ROUND TABLE, GREAT FALLS.

in which they were produced by the round tables, the additional cost of fluxing the silica contained in the canvas-deck concentrate, together with the additional cost of loading, transporting, and unloading, would probably more than offset the advantage gained by the extra recovery of 5 per cent. of copper made by the deck.

As, however, the round tables were used as roughing tables, the concentrate produced being cleaned up on finishing tables, the canvas-deck table would appear to have some advantages in making a 5 per cent.

higher recovery of copper than the cement deck. It is noted, however, that the assay value of the total tailing produced by the canvas deck is slightly higher than that of the similar product of the cement deck, the extra recovery of copper made by the canvas deck being traceable to a slightly richer feed.

The assay value of the feed-side tailing produced by the canvas deck is slightly higher than that of the cement deck, while for the wash-side tailing the copper content is lower in the case of the canvas deck.

Test Results Obtained on Steel-Frame Cement and Canvas Deck Surfaces

	Kind of Deck Surface	
	Cement	Canvas
Time to make one revolution, minutes	19	19
Slope of table deck in inches per foot	1.2	1.13
Tons of dry solids fed per 24 hr	8.3	8.3
Assay per cent. copper in feed	3.36	3.46
Density of feed in grams per gallon	374	390
Tons of concentrate produced per 24 hr	2.9	4.3
Ratio of concentration, tons into one	2.9	1.9
Assay per cent. copper in concentrate	7.84	5.79
Assay per cent. insoluble in concentrate	67.3	74.7
Assay per cent. FeO in concentrate	10.5	7.9
Per cent. of copper fed, recovered in concentrate	82.01	86.98
Gallons of fresh water added per 24 hr	10,112	12,589
Gallons of fresh water added per ton fed	1,218	1,517
Gallons of fresh water added per ton concentrate	3,487	2,928
Tons of north-side tailing produced per 24 hr	3.2	2.2
Assay per cent. copper of above	0.95	0.98
Per cent. of feed solids in north-side tailing	38.16	26.45
Per cent. of feed copper in north-side tailing	10.79	7.46
Tons of south-side tailing produced per 24 hr	2.2	1.8
Assay per cent. copper of above	0.91	0.88
Per cent. of feed solids in south-side tailing	26.69	21.57
Per cent. of feed copper in south-side tailing	7.19	5.56
Tons of total tailing produced per 24 hr	5.4	4.0
Assay per cent. copper of above	0.93	0.94
Per cent. of feed solids in total tailing	64.85	48.02
Per cent. of feed copper in total tailing	17.99	13.02

From the results obtained, it was the opinion of the writer that if the slope of the canvas deck had been the same as that of the cement deck, the advantage gained by the canvas deck in recovering an additional 5 per cent. of copper would have been wiped out in an increased tailing loss without very materially benefiting the grade of concentrate produced It was, therefore, considered that the cement deck was the equal of the canvas deck in regard to its ability to recover copper mineral and its superior in producing a cleaner grade of concentrate, in using less water, and in possessing a longer life.

Our experience with canvas deck surfaces has shown that the canvas has a comparatively short life, becoming rotten after about nine months' use. During September, 1910, a test had been made on a canvas-deck round table at the slime plant, the canvas being supported on a wooden deck surface with a slope of 1.25 in. per foot; in this case the canvas deck made a cleaner concentrate than the steel-frame canvas deck with a slope of 1.13 in. per foot, but the recovery of copper made was much lower. The comparative results of both tests are shown below.

Data from Tests Made on Canvas-Deck Round Tables

	Steel-frame Canvas Deck	Wood-frame Canvas Deck
Slope of deck surface in inches per foot	1.13	1.25
Time to make one revolution, minutes	19	19
Tons dry solids treated per 24 hr	8.3	7.6
Assay per cent. copper in feed	3.46	2.23
Density of feed in grams per gallon	390	409
Tons of concentrate produced per 24 hr	4.3	1.5
Ratio of concentration, tons into one	1.9	5.2
Assay per cent. copper in concentrate	5.79	7.33
Assay per cent. insoluble in concentrate	74.7	66.0
Assay per cent. FeO in concentrate	7.9	10.7
Per cent. of copper fed, recovered in concentrate	86.98	63.42
Gallons of fresh water added per 24 hr	12,589	21,531
Gallons of fresh water added per ton fed	1,517	2,833
Gallons of fresh water added per ton of concentrate	2,928	14,354
Tons of north-side tailing produced per 24 hr	2.2	2.97
Assay per cent. copper of above	0.98	0.94
Per cent. of feed solids in north-side tailing	26.45	39.75
Per cent. of feed copper in north-side tailing	7.46	16.92
Tons of south-side tailing produced per 24 hr	1.8	3.17
Assay per cent. copper of above	0.88	1.07
Per cent. of feed solids in south-side tailing	21.57	42.30
Per cent. of feed copper in south-side tailing	5.56	20.54
Tons of total tailing produced per 24 hr	4.0	6.1
Assay per cent. copper of above	0.94	1.01
Per cent. of feed solids in total tailing	48.02	82.05
Per cent. of feed copper in total tailing	13.02	37.46

Tailing products called north and south side tailings when the round tables are operated as roughing tables, as far as the side of the table from which they are delivered is concerned, correspond to feed and wash side tailings of finishing-table practice.

The foregoing comparison is given to show that a clean grade of concentrate can be produced from the canvas deck with an increase of the deck slope. The recovery of copper made by the wood-frame canvas

deck, in producing a concentrate similar in character to the concentrate produced by the steel-frame cement deck, is 18.6 per cent. lower than the recovery made by the steel-frame cement deck. This great difference in recovery of copper is partly accounted for by the fact that the canvas deck treated a lower grade of feed than the cement deck. The canvas deck, however, in treating a slightly lower tonnage of a lower grade of feed, made a tailing which assayed higher in copper than the tailing produced by the cement deck.

Referring again to the test results obtained on the steel-frame cement deck and the steel-frame canvas deck. Eight sprays were used to wash off the concentrate from the canvas deck and two sprays on the cement deck; the spray nozzles on the canvas deck, however, had smaller diameters than those used on the cement deck.

A great deal more solid material settled out on the canvas deck than on the cement deck, during the test when a similar tonnage was treated on each kind of deck, the concentrate on the canvas deck amounting to 51.98 per cent. of the total solids fed, while the cement-deck concentrate amounted to only 35.15 per cent. of the total solids fed.

The results of the test indicate that given the same slope of deck surface, a higher grade of concentrate can be produced on a cement deck surface without any greater tailing loss than is incurred on a canvas surface.

In May, 1911, the speed of the tables was increased from 1 rev. in 19 min. to 1 rev. in 16 min., the object of the change chiefly concerning an improvement in the work of the canvas-deck table, it being expected that this change would enable this deck to produce a cleaner grade of concentrate with less tailing loss. The tables were tested under the new speed conditions but the results did not show that any advantage was gained.

The work of the steel-frame cement-deck table will be again referred to in connection with tests made on conoidal deck surfaces.

Some of the principal data obtained from the sizing tests made on the feed and products of the steel-frame wood and canvas decks in the above test are given below:

Feed to tables:	Steel-frame Decks	
	Cement	Canvas
Per cent. of total solids on 200 mesh (0.07 mm.)	4.34	6.54
Assay per cent. copper of above	0.81	0.85
Per cent. of total copper on 200 mesh	1.13	1.74
Per cent. of total solids through 200 mesh	95.66	93.46
Assay per cent. copper of solids through 200 mesh	2.98	2.89
Per cent. of total copper through 200 mesh	98.87	98.26
Assay per cent. copper of total feed	2.67	2.76

	Steel-frame Decks	
	Cement	Canvas
Concentrate produced:		
Per cent. of total solids on 200 mesh	2.05	3.97
Assay per cent. copper of above	1.35	2.10
Per cent. of total copper on 200 mesh	0.41	1.52
Per cent. of total solids through 200 mesh	97.95	96.03
Assay per cent. copper of solids through 200 mesh	6.95	4.90
Per cent. of total copper through 200 mesh	99.59	98.48
Assay per cent. copper of total concentrate	6.95	4.80
North-side tailing:		
Per cent. of total solids on 200 mesh	5.36	9.13
Assay per cent. copper of above	0.28	0.33
Per cent. of total copper on 200 mesh	1.30	2.70
Per cent. of total solids through 200 mesh	94.64	90.87
Assay per cent. copper of solids through 200 mesh	1.20	1.12
Per cent. of total copper through 200 mesh	98.70	97.30
Assay per cent. copper of total north-side tailing	1.20	1.05
South-side tailing:		
Per cent. of total solids on 200 mesh	8.43	12.26
Assay per cent. copper of above	0.36	0.66
Per cent. of total copper on 200 mesh	3.48	7.13
Per cent. of total solids through 200 mesh	91.57	87.74
Assay per cent. copper of solids through 200 mesh	1.02	1.00
Per cent. of total copper through 200 mesh	96.52	92.87
Assay per cent. copper of total south-side tailing	0.99	0.93

From the figures above it can be seen that the original feed to the round tables contained from 4.3 to 6.5 per cent. of material coarser than 200 mesh.

The figures given below show the grade of tailing made by the round tables from the material coarser than 200 mesh.

Name of Tailing	Total Pounds per 24 Hr.	Per Cent. on 200 Mesh	Pounds on 200 Mesh	Assay Per Cent. Copper	Pounds Copper on 200 Mesh
		Steel-Frame Cement-Deck Table			
North	4,941	5.36	265	0.28	0.74
South	3,693	8.43	311	0.36	1.12
Total	8,634	6.67	576	0.32	1.86
		Steel-Frame Canvas-Deck Table			
North	3,209	9.13	293	0.33	0.97
South	5,765	12.26	707	0.66	4.67
Total	8,974	11.26	1,000	0.56	5.64

The figures above show that with only 4.34 per cent. of on 200-mesh material contained in the feed to the table, the cement deck makes a very satisfactory tailing, assaying only 0.32 per cent. copper, this figure being comparable with a result of 0.49 per cent. copper obtained in a previous test when the feed to the deck contained 9.09 per cent. of on 200-mesh material. The on 200-mesh tailing produced by the canvas deck was somewhat higher, the amount of on 200-mesh material in the feed being also somewhat higher.

From the results of the above and previous sifting tests the writer concluded that the amount of on 200-mesh material in slime-plant feed should not exceed 4 per cent., and that the work of the tables could be improved if this amount were still further reduced.

Conoidal Deck Round Tables

In May, 1912, the question of the use of conoidal deck round tables for slimes again came up for discussion, the subject being brought up by Prof. Henry S. Munroe of Columbia University.

Professor Munroe stated that his object in making the round-table deck conoidal instead of conical, was to make the movement of the pulp on the deck, and the concentration conditions, uniform, in spite of the divergence of the water on the deck as it flows from the center toward the circumference; he, therefore, adopted the plan of gradually increasing the slope of the table from the center to the circumference. Referring to the fact that as the stream of pulp on the table spreads out, the film becomes shallower and the transporting power becomes less, Professor Munroe stated that the most advantageous way to overcome this was to change the deck surface as described above. He stated that the frame and steel top of the round table experimented with at Columbia had a uniform slope of 4°.

The surface of the cement, however, was rounded so that the inclination varied from 3° at the center to 5° at the circumference. The table had a diameter of 18 ft. 5 in., so that the change of inclination was very gradual, something less than a quarter of a degree to the foot, and at the middle point of the profile the curved surface was only about ½ in. above a straight line. This table was designed for the treatment of medium-fine slime, through about 0.15 mm.; for the very fine slime treated at Great Falls, Professor Munroe recommended a slightly steeper slope.

The first conoidal deck surfaces tried out at Great Falls were installed by laying a conoidal cement surface on three of the wooden round-table decks which were still operating in section No. 5 of the concentrator. The tables, therefore, received the regular sand-slime feed of this section, which, as was mentioned earlier in this paper, was very poorly classified, containing grains ranging in size from 0.5 mm. to the very finest slime.

A rough finish and a smooth finish deck surface were tested out under high, medium, and low tonnage conditions. The conoidal deck surface was laid on so that the chord of the arc made an angle of 4° 30′ (1 in. per foot) with the horizontal.

A large number of tests were made and the tables did good work considering the class of feed treated; a good recovery of copper was made in a fair grade of concentrate but no waste tailing could be made by the tables when handling this class of feed. As the flow sheet for future use in this section of the concentrator had already been decided upon, and as this practice could hardly be improved by the use of round tables unless a very close sorting of the feed was resorted to, it was determined to try out the conoidal deck surfaces in the slime plant, where the field was still open for any suggested improvements in round-table practice.

Results with Sand-Slime Feed on Conoidal Deck

Before presenting data on the use of conoidal round-table deck surfaces in the slime plant, a few of the test results obtained from the operation of the tables in the sand-slime section of the concentrator will be presented to illustrate the character of the work done by the tables.

Test Results from Conoidal Deck Round Tables

	Rough Finish Deck Surface		
Character of feed treated (size)	Medium	Fine	Fine
Rate of feed	High	Medium	Low
Rate of feed in tons per 24 hr	31.500	12.349	4.826
Density of feed in grams per gallon	693	358	259
Density of feed, per cent. of solids in pulp	16.48	9.0	7.14
Assay per cent. copper in feed	4.25	3.37	4.23
Tons of concentrate produced per 24 hr	3.823	1.554	0.675
Density of concentrate in grams per gallon	126	45	20
Density of concentrate, per cent. solids in pulp	3.25	1.20	0.55
Assay per cent. copper in concentrate	18.90	16.64	19.06
Assay per cent. insoluble in concentrate	18.4	18.4	31.8
Assay per cent. FeO in concentrate	31.2	29.4	20.4
Per cent. of copper fed, recovered in concentrate	54.02	62.14	62.99
Ratio of concentration, tons into one	8.24	7.95	7.15
Feed-side tailing solids, per cent. of total solids fed	63.57	14.11	27.73
Feed-side tailing, assay per cent. copper	2.09	2.17	2.14
Feed-side tailing, copper per cent. of total copper fed	31.29	9.13	13.97
Wash-side tailing solids, per cent. of total solids fed	24.29	73.31	58.28
Wash-side tailing, assay per cent. copper	2.57	1.32	1.67

Wash-side tailing, copper, per cent. of total copper fed	14.69	28.73	23.04
Gallons of dressing water used per 24 hr	13,887	25,133	18,077
Gallons of dressing water used per ton of feed	440	2,035	3,746
Gallons of spray water used per 24 hr	25,640	30,624	29,334
Gallons of spray water used per ton of feed	814	2,480	6,078
Pounds of feed water per minute per foot of total circumference	4.15	2.88	1.70
Pounds of dressing water per minute per foot of total circumference	1.39	2.60	1.82
Pounds of spray water per minute per foot of total circumference	2.86	3.32	3.18
Revolutions of table per minute	2	2	2
Number of sprays operating	8	8	8

An explanation of the "Character of feed treated" in the preceding tests is as follows: Medium feed is a mixture of all spigot discharges of the feed tank; fine feed is the product of the last three spigot discharges of the feed tank. The spray water used for washing off the concentrate was afterward greatly reduced by using five spray nozzles of smaller diameter instead of eight.

The high-tonnage test above mentioned was run to produce a slightly poorer grade of concentrate than we had been getting in previous tests on high-tonnage feeds and in this respect the test was successful, the change in the concentrate being practically the only variable with the exception of a slight drop in the rate of feed. By increasing the insoluble content of the concentrate 3 per cent. the recovery of copper in concentrate was raised 10 per cent.

In the medium-tonnage test made on a fine feed a very clean grade of concentrate was produced and a fair recovery of copper made, the recovery being as good and the grade of concentrate better than in the low-tonnage test on the same class of feed.

	Smooth Finish Deck Surface		
Character of feed treated	Medium	Medium	Fine
Rate of feed	High	Medium	Low
Rate of feed in tons per 24 hr	20.197	14.039	6.261
Density of feed in grams per gallon	452	271	292
Density of feed, per cent. of solids in pulp	11.93	7.15	7.71
Assay per cent. copper in feed	4.86	3.23	2.63
Tons of concentrate produced per 24 hr	3.493	1.661	0.788
Density of concentrate in grams per gallon	1.39	90.0	44.0
Density of concentrate, per cent. solids in pulp	3.64	2.36	1.19
Assay per cent. copper in concentrate	17.0	16.08	13.9
Assay per cent. insoluble in concentrate	29.6	26.0	35.3
Assay per cent. FeO in concentrate	26.4	27.50	25.6
Per cent. of copper fed, recovered in concentrate	60.52	58.81	66.56
Ratio of concentration, tons into one	5.78	8.45	7.95

Feed-side tailing solids, per cent. of total solids fed	42.81	56.11	8.29
Feed-side tailing, assay per cent. copper	2.2	1.43	1.93
Feed-side tailing copper, per cent. of total copper fed	19.36	24.78	6.08
Wash-side tailing solids, per cent. of total solids fed	39.90	32.06	79.12
Wash-side tailing, assay per cent. copper	2.45	1.65	0.91
Wash-side tailing copper, per cent. of total copper fed	20.12	16.41	27.36
Gallons of dressing water used per 24 hr	13,150	19,992	21,151
Gallons of dressing water used per ton of feed	651	1,424	3,378
Gallons of spray water used per 24 hr	22,106	16,447	15,598
Gallons of spray water used per ton of feed	1,094	1,171	2,491
Pounds of feed water per minute per foot of total circumference	3.88	4.74	1.95
Pounds of dressing water per minute per foot of total circumference	1.34	2.15	2.11
Pounds of spray water per minute per foot of total circumference	2.40	1.79	1.69
Revolutions of table per minute	2	2	2
Number of sprays operating	8	8	8

The high-tonnage test results showed a fair recovery of copper in a fair grade of concentrate, the recovery of copper being higher than in the medium-tonnage test immediately following although the grade of concentrate is not so good. The lower copper recovery of the medium-tonnage test is due to the fact that a lower grade of feed was treated, and the cleaner concentrate to the greater quantity of dressing water used. In the low-tonnage test a higher recovery of copper was made, but the grade of concentrate produced was much poorer, the grade of the feed treated being much lower.

In closing a report on the above tests the writer suggested that the conoidal deck round table be given a trial in the slime plant as it presented possibilities of development into a useful table for the treatment of very fine slime.

In the slime-plant practice prevailing at this time, the conical deck round tables were being used as roughing tables, the tables being fed over their entire surface, with the exception of the part reserved for washing off concentrate, and discharging tailing material from the total perimeter with the above exception. The round-table decks used had a slope of 1.25 in. per foot. This slope, with a favorable density of feed, had been proved to give a maximum recovery of free-mineral grains, the coarser grains being recovered on the upper part of the deck and the very minute grains toward the perimeter of the deck. In recovering the very minute mineral grains, however, the tables also recovered a large amount of gangue material, this having been determined by qualitative experiments carried out at the suggestion of C. W. Goodale, in which samples

of the concentrate on the deck surface were taken for each foot from the center of the table to the perimeter. These experiments, which will be described later, developed the fact that the concentrate deposited on the lower half of the deck was of a very low grade, and within 12 in. of the perimeter of the deck was practically a tailing material. In the opinion of the writer, the conoidal deck with its very gradual increase in slope presented possibilities for overcoming the above defect. The first Munroe conoidal deck round table with cement surface (Fig. 2) was installed in the slime plant and ready for operation on Dec. 3, 1912. The

Deck surface, cement mortar (2 parts sand to 1 part cement). Slope of chord of arc = 1.25 in. per foot. Diameter of deck, 17 ft. Table operating to produce a finished concentrate, a middling for re-treatment, and a low-grade tailing product. One revolution in 16 min.

FIG. 2.—CONOIDAL DECK ROUND TABLE, GREAT FALLS. CONOIDAL DECK NO. 1.

table was installed with a total slope of 6° (1.25 in. per foot), as, from our preliminary experiments made in section No. 5, this slope seemed to be the most satisfactory. Hence, the chord of the arc of the conoidal deck made the same angle with the horizontal as the surface of the conical deck.

The cement mortar used for the deck surface consisted of two parts of sand to one of cement, the surface being given a rough finish similar to the surface of the steel-frame conical decks. The conoidal deck was laid on one of the wooden decks in the slime plant, the following figures representing the cost of the work and the weight of materials used:

Cost		Weight of Material Used	Pounds.
Sand, 2,890 lb. @ 17c. per ton	$0.25	Cement, 14 sacks @ 93 lb. per sack	1,302
Cement, 14 sacks @ 70c. per sack	9.80	Sand at 3,000 lb. per cubic yard	2,890
Laborers, 4 days @ $3.25 per day	13.00		
Drayage	2.00	Total	4,192
Total	$25.05		

The total slope of 6° which was given to the conoidal deck surface gave the following slope values for each foot of deck surface from the center of the table to the perimeter.

	Degrees and Minutes to Nearest Minute	Inches per Foot to Nearest $\frac{1}{8}$ in.
First foot	4° 30′ to 4° 53′	$\frac{1}{8}$ to 1
Second foot	4° 53′ to 5° 15′	1 to $1\frac{1}{16}$
Third foot	5° 15′ to 5° 38′	$1\frac{1}{16}$ to $1\frac{1}{8}$
Fourth foot	5° 38′ to 6° 0′	$1\frac{1}{8}$ to $1\frac{1}{4}$
Fifth foot	6° 0′ to 6° 23′	$1\frac{1}{4}$ to $1\frac{5}{16}$
Sixth foot	6° 23′ to 6° 45′	$1\frac{5}{16}$ to $1\frac{3}{8}$
Seventh foot	6° 45′ to 7° 8′	$1\frac{3}{8}$ to $1\frac{1}{2}$
Eighth foot	7° 8′ to 7° 30′	$1\frac{1}{2}$ to $1\frac{5}{8}$

The deck had a diameter of 17 ft. and an available radial width of working surface of 7 ft. 3 in. The table was timed to make 1 rev. in 20.5 min., or 70.24 rev. per 24 hr.

This table was tested against a steel-frame cement-deck table of the following specifications:

The table tested was the upper deck of the steel-frame four-deck conical table. This deck had a slope of 1.25 in. per foot and the cement surface of the deck had a similar finish to that of the conoidal deck. The deck had a diameter of 18 ft. and an available radial width of working surface of 7 ft. 9 in. The table made 1 rev. in 16 min., or 90 rev. per 24 hr.

During the months of December, 1912, and January, 1913, a large number of comparative tests were carried out on the conoidal and the conical decks, the details of which, although very interesting, the writer is obliged to omit in order to keep the length of this paper within reasonable limits. During this period the tables were operated both as roughing tables producing a waste tailing product and a rough concentrate for re-treatment on Deister tables, and as finishing tables producing a waste tailing, a middling for re-treatment on Deister tables, and a finished concentrate, both systems of operation being carried out under various feed-density conditions and various tonnage conditions.

The operation of the tables to produce a finished concentrate, a middling for re-treatment, and a waste tailing resulted from a consideration of

the assay results obtained on samples of rough concentrate taken at regular intervals between the center of the deck and its perimeter. From these results it was evident that a considerable percentage of the rough concentrate could be recovered in a concentrate which would be equal in grade to the finished concentrate turned out by the plant after re-treating the total round-table concentrate on Deister tables.

The adoption of this latter system would mean a reduction of the load on the Deister tables and a lowering of the re-treatment loss.

The middling for re-treatment was washed off the lower 3 ft. 6 in. to 4 ft. 6 in. of the deck, by a jet of water, into a separate launder which immediately preceded the finished-concentrate launder. The nozzle used for directing the jet of water was attached to a flexible connection so that the amount of material produced for re-treatment could be varied as desired.

Another method used for producing a clean concentrate and a middling for re-treatment was as follows:

The corner of the feed box nearest to and preceding the concentrate wash-off jets was partitioned off and used as a small wash-water box, fresh water being added at this point and allowed to flow over the concentrate just before it reached the wash-off sprays; the material washed off the deck in this manner was sent to the Deister tables for re-treatment. Either method gives very satisfactory results.

Conclusions from Preliminary Tests on Conoidal and Conical Decks

It was evident from the results obtained both in the roughing tests and in the finishing tests that a conoidal deck with an overall slope of 1.25 in. per foot and a conical deck with an overall slope of 1.25 in. per foot will not yield similar end results under similar feed conditions, the conoidal deck requiring a denser feed pulp.

The tabular comparison (p. 447) illustrates very clearly that if we feed a dense pulp to the conoidal deck and a more dilute pulp to the conical deck, practically similar end results may be expected; this refers to the tables as installed at this time. To produce similar end results it was found that the feed to the conoidal deck must be thickened to a density about 75 per cent. greater than that of the feed to the conical deck.

Since it might not be found advisable to thicken the table feed to a density of 500 g. of solids per gallon of pulp (12.2 per cent. solids) the next best thing to do would be to flatten the conoidal deck; that is, to reduce the overall slope from 1.25 in. per foot to 1.125 or 1 in. per foot, the latter slope, in the opinion of the writer, being the extreme limit in this direction for the class of feed which was being treated in the slime plant at this time.

Comparative Results of Roughing Tests on Conoidal and Conical Decks

	Name of Deck				
	Conoidal		Conical		
	A	H	B	C	D
Tons dry solids treated per 24 hr.....	7.979	6.586	6.146	6.913	8.649
Density of feed in grams per gallon....	469	501	287	295	280
Assay per cent. copper in feed.....	2.69	2.50	2.34	2.44	2.20
Tons concentrate produced per 24 hr..	2.243	2.180	1.711	1.890	2.438
Assay per cent. copper in concentrate..	7.11	5.53	6.05	6.88	5.84
Assay per cent. insoluble in concentrate.	67.0	73.1	71.7	64.5	71.2
Assay per cent. FeO in concentrate....	11.8	9.2	9.6	12.8	10.4
Per cent. of copper fed, recovered in concentrate.....	74.36	73.03	71.87	76.92	74.80
Ratio of concentration, tons into one..	3.557	3.021	3.592	3.658	3.547
Ratio of enrichment (copper).....	2.64	1.37	2.59	2.8	2.65
Tons north-side tailing produced per 24 hr.....	2.071	2.086	2.496	2.062	3.183
Assay per cent. copper.....	1.01	0.98	0.96	0.75	0.77
Tons south-side tailing produced per 24 hr.....	3.664	2.320	1.938	2.960	3.029
Assay per cent. copper.....	0.93	1.03	0.85	0.79	0.78
Tons total tailing produced per 24 hr..	5.735	4.406	4.434	5.022	6.212
Assay per cent. copper.....	0.95	1.01	0.91	0.78	0.77
Per cent. of copper fed, lost in north-side tailing.....	9.79	12.42	16.67	9.17	12.86
Per cent. of copper fed, lost in south-side tailing.....	15.85	14.55	11.46	13.91	12.34
Per cent. of copper fed, lost in total tailing.....	25.64	26.97	28.13	23.08	25.20
Gallons of fresh water used per 24 hr..	8,590	10,025	13,821	13,122	11,891
Gallons of fresh water used per ton fed	1,076	1,522	2,249	1,898	1,375

NOTE.—North-side tailing = tailing usually produced from feed side. South-side tailing = tailing usually produced from wash side.

Under the "deck slope" conditions of the tests, with feed conditions similar, *the tables operating to produce a rough concentrate and a waste tailing*, the following end results are obtained:

(*a*) The conoidal deck will deliver from 17 to 25 per cent. less of the feed solids in the concentrate than the conical deck.

(*b*) The concentrate produced by the conoidal deck will assay from 1.5 to 3.0 per cent. higher in copper, 6 to 12 per cent. less in insoluble and 3 to 5 per cent. higher in iron than the concentrate produced by the conical deck.

(*c*) The concentrate produced by the conoidal deck will contain from 6.5 to 8.0 per cent. less of the copper fed to the table than the concentrate produced by the conical deck.

The same relative conditions maintain for any other method of operating the tables, hence, if we wish to operate conoidal and conical decks in parallel under similar feed conditions to produce similar end results: For any given density of feed the conoidal decks must be installed with a less overall slope than the conical decks.

It was considered possible that by flattening the chord of the conoidal deck by $\frac{1}{16}$ in. per foot at a time, using an experimental deck section, a slope might be found which, while giving a recovery equal to that made by the conical deck, would produce a better grade of concentrate than is produced by the conical deck. The writer considered that the building of a conoidal deck for regular operation with an overall slope of 1.125 in. per foot was justifiable and this was done later.

A conoidal deck with an overall slope of 1.0 in. per foot was first constructed, this table doing very satisfactory work if the table feed did not get above medium density and did not contain more than 2 or 3 per cent. of fine sand. A preliminary test was made on this table on Feb. 28, 1913, when the feed to the table had a density of 468 g. of solids per gallon of pulp; the density of feed was too high, the table making an extraction of 85.6 per cent. of the copper fed, in a concentrate assaying 4.12 per cent. copper. Other tests were made later on this table using less dense feed with better results, as presented in the following pages.

In a general way it would seem that it makes very little difference which type of deck is used as long as we use a slope on each type of deck which is most suitable for the average density of feed handled at the slime plant. This conclusion is subject, however, to results to be obtained by using a flatter conoidal deck, as stated on a preceding page, as it was considered possible that the recovery made by the conical deck might be reached in a better grade of concentrate.

In tests *A* and *H*, when a high density of feed was used, averaging 480 g. of solids per gallon of pulp, the writer considered that the work of the conoidal deck was superior to that of the conical deck in spite of the fact that the former made a recovery of copper lower by 6.82 per cent. than the other, 73.78 against 80.70 per cent. (averages of tests *A* and *H*). (*A* and *H* tests on conical deck are not included in the preceding table, but data on tailings are given in the following table.)

This was on account of the fact that the conoidal deck sent to tailings 20.4 per cent. more of the feed solids than the conical deck, the tailings having similar assay values. This means that the conoidal deck produced much less material for re-treatment, the additional loss from this deck being entirely due to additional tailing production and this tailing product being of a lower grade than the tailing made in the re-treatment system. An examination of the following figures will make this point clear.

Test	Pounds Total Tailing Produced per 24 Hr.	Per Cent. of Solids in Feed	Assay Per Cent. Copper	Pounds Copper per 24 Hr.	Per Cent. of Copper in Feed
		Conoidal Deck			
A	11,472	71.89	0.96	110	25.64
H	8,812	66.90	1.01	89	26.97
Total and Average	20,284	69.63	0.98	199	26.22
		Conical Deck			
A	7,067	48.10	1.05	74	19.07
H	7,133	50.41	0.98	70	19.55
Total and Average	14,200	49.23	1.01	144	19.30

The results obtained from both the conoidal and the conical deck indicated that considerable improvement could be made in the round-table practice if the slimes were roughly classified preceding table treatment.

The main difference in the work of the two tables under discussion, both having an overall slope of 1.25 in. per foot, may be explained as follows:

The conical deck, as installed, is capable of retaining upon its surface a considerable percentage of the very finest mineral grains contained in the slime, but in doing so also retains a large percentage of the fine sand, the result being that a concentrate high in insoluble material is produced and a high recovery of copper made.

The conoidal deck, as installed, cannot retain upon its surface a high percentage of the very finest mineral grains, but in losing these fine mineral grains the deck also gets rid of a high percentage of the fine sand, the result being that a clean grade of concentrate is produced, but the recovery of copper made is lower than that made by the conical deck.

From the behavior of the two types of table deck under varying feed densities it would at first seem that the work of the two types of tables under constant feed conditions could be made similar either by increasing the slope of the conical deck or by flattening the conoidal deck. The test data on the second conoidal deck installed in the slime plant will threw some further light on this subject.

Professor Munroe's idea in adopting the conoidal deck surface, as previously stated, was to make the movement of the pulp and the concentration conditions uniform in spite of the divergence of the water as it flows from the center toward the circumference, which was accomplished by gradually increasing the slope of the table and, therefore, the speed of the film of water. This condition allows the use of a flatter slope than would be practical on a conical deck.

A heavy deposition occurs at the center of the table, but with a slope

at this point of 4° 30′ (⅛ in. per foot), which is the slope of the table tested, this deposition has been very clean mineral. The writer considers that it would be safe to make the slope at the center of the table 4° or possibly a shade less if the density of the feed is kept at about 350 g. of solids per gallon of pulp (9.25 per cent. solids). With a 3° slope at the center of the table, as suggested by Professor Munroe, a good deal of fine sand is deposited with the mineral if the slime contains 2 or 3 per cent. of on 200-mesh material, and it is difficult to clean this concentrate with dressing water.

The second conoidal deck installed at the slime plant, known as conoidal deck No. 2, was given an overall slope of 4° 45′, or 1 in. per foot, this giving a deck surface inclination varying from 3° 30′ at the center to 6° 15′ at the circumference, or very nearly the slope which is recommended by Professor Munroe.

The above deck was tested during the period April 26 to 29, inclusive, 1913, in parallel with conoidal deck No. 1, the steel-frame conical deck, and the step-deck previously described in this paper.

During the test an endeavor was made to keep the tonnage fed to each table and the density of the feed uniform.

The comparative results obtained from the tests made on the four tables are shown below:

Comparison of Data Obtained on Four Round Tables

	Conoidal Deck No. 1	Conoidal Deck No. 2	Step Deck	Conical Deck
Feed:				
Gallons of feed pulp per 24 hr	18,272	18,362	17,705	17,106
Tons dry solids in feed per 24 hr	9.909	9.736	9.700	9.296
Density of feed in grams per gallon	492	481	497	493
Density of feed, per cent. solids	12.0	11.5	12.1	12.0
Assay per cent. copper in feed	2.91	2.94	2.74	2.71
Pounds of copper fed per 24 hr	576	572	532	503
Concentrate No. 1:				
Gallons per 24 hr	9,177	11,562	7,957	14,694
Tons dry solids per 24 hr	1.379	2.940	1.604	1.403
Density in grams per gallon	136	231	183	87
Density in per cent. solids	2.9	5.3	4.3	2.2
Per cent. of solids in feed	13.91	30.19	16.53	15.09
Assay per cent. copper	13.46	7.26	8.42	10.87
Assay per cent. insoluble	37.8	64.9	59.3	48.7
Assay per cent. FeO	23.6	12.4	14.9	18.8
Pounds of copper per 24 hr	371	427	270	305
Per cent. of copper in feed	64.41	74.65	50.75	60.64
Concentrate No. 2 (middling):				
Gallons per 24 hr	2,048	4,543	10,274	6,205
Tons dry solids per 24 hr	0.150	0.680	0.637	0.952
Density in grams per gallon	66	136	56	139

	Conoidal Deck No. 1	Conoidal Deck No. 2	Step Deck	Conical Deck
Concentrate No. 2 (middling)—*Continued:*				
Density in per cent. solids (approximate)	1.7	3.5	1.6	3.6
Per cent. of solids in feed	1.51	6.99	6.55	10.23
Assay per cent. copper	4.35	1.91	7.16	2.99
Assay per cent. insoluble	76.6	88.1	65.3	84.0
Assay per cent. FeO	7.0	2.8	11.8	4.3
Pounds of copper per 24 hr	13	26	91	57
Per cent. of copper in feed	2.26	4.55	17.11	11.33
Total concentrate:				
Gallons per 24 hr	11,225	16,105	18,231	20,899
Tons dry solids per 24 hr	1.528	3.620	2.239	2.354
Density in grams per gallon	123	204	111	102
Density in per cent. solids (approximate)	3.20	5.1	2.9	2.7
Per cent. of solids in feed	15.42	37.18	23.08	25.32
Assay per cent. copper	12.56	6.26	8.06	7.69
Assay per cent. insoluble	41.6	69.3	61.0	63.0
Assay per cent. FeO	22.0	10.6	14.0	12.9
Pounds of copper per 24 hr	384	453	361	362
Per cent. of copper in feed	66.67	79.20	67.86	71.97
North-side tailing:				
Gallons per 24 hr	10,208	8,995	9,212	11,502
Tons dry solids per 24 hr	4.056	2.726	2.536	3.616
Density in grams per gallon	360	275	250	285
Density in per cent. solids (approximate)	9.0	7.0	6.35	7.3
Per cent. of solids in feed	40.92	28.0	26.14	38.90
Assay per cent. copper	1.08	1.08	1.16	1.05
Pounds of copper per 24 hr	88	59	59	76
Per cent. of copper in feed	15.28	10.31	11.09	15.11
South-side tailing:				
Gallons per 24 hr	10,801	9,009	13,838	9,606
Tons dry solids per 24 hr	4.326	3.390	4.925	3.326
Density in grams per gallon	363	341	323	314
Density in per cent. solids (approximate)	9.0	8.5	8.1	7.8
Per cent. of solids in feed	43.66	34.82	50.78	35.78
Assay per cent. copper	1.20	0.88	1.14	0.98
Pounds of copper per 24 hr	104	60	112	65
Per cent. of copper in feed	18.05	10.49	21.05	12.92
Total tailing:				
Gallons per 24 hr	21,009	18,004	23,050	21,108
Tons dry solids per 24 hr	8.382	6.116	7.461	6.942
Density in grams per gallon	362	308	294	298
Density in per cent. solids (approximate)	9.0	7.8	7.5	7.5
Per cent. of solids in feed	84.58	62.82	76.92	74.68
Assay per cent. copper	1.14	0.97	1.15	1.01
Pounds of copper per 24 hr	192	119	171	141
Per cent. of copper in feed	33.33	20.80	32.14	28.03

Discussion of Results of Four-Day Test on Conical and Conoidal Decks

The four-day test results place the decks in the following order in regard to efficiency.

In Regard to Copper Recovery	Based on Total Concentrate Per Cent. Recovery	In Regard to Grade of Concentrate
Conoidal deck No. 2	79.20	Conoidal deck No. 1, 12.6 per cent. Cu; 41.6 per cent. insol.; 22.0 per cent. FeO.
Conical deck	71.97	Step deck, 8.1 per cent. Cu; 61.0 per cent. insol.; 12.9 per cent. FeO.
Step deck	67.86	Conical deck, 7.7 per cent. Cu; 63.0 per cent. insol.; 12.9 per cent. FeO.
Conoidal deck No. 1	66.67	Conoidal deck No. 2, 6.3 per cent. Cu; 69.3 per cent. insol.; 10.6 per cent. FeO.
In Regard to Copper Recovery	Based on No. 1 Concentrate Per Cent. Recovery	In Regard to Grade of Concentrate
Conoidal deck No. 2	74.65	Conoidal deck No. 1, 13.5 per cent. Cu; 37.8 per cent. insol.; 23.6 per cent. FeO.
Conoidal deck No. 1	64.41	Conical deck, 10.9 per cent. Cu; 47.8 per cent. insol.; 18.8 per cent. FeO.
Conical deck	60.64	Step deck, 8.4 per cent. Cu; 59.3 per cent. insol.; 14.9 per cent. FeO.
Step deck	50.75	Conoidal deck No. 2, 7.3 per cent. Cu; 64.9 per cent., insol.; 12.4 per cent. FeO.

Before discussing the above results some figures will be given showing the fresh water added on the tables during the test period.

	Conoidal Deck No. 1	Conoidal Deck No. 2	Step Deck	Conical Deck
Gallons of fresh water used per 24 hr	13,960	15,750	23,580	24,900
Gallons of fresh water used per ton fed	1,410	1,620	2,430	2,680
Gallons of No. 1 concentrate produced	9,180	11,560	7,960	14,690
Gallons of No. 2 concentrate produced	2,050	4,540	10,270	6,210
Per cent. of added water to wash off No. 1 concentrate	66.0	73.0	34.0	59.0
Per cent. of added water used as dressing water.	34.0	27.0	66.0	41.0

Most of the water used as dressing water finds its way into No. 2 concentrate (middling), some of it, however, going to south-side tailing and some to concentrate No. 1.

The order of the decks as regards efficiency in copper recovery made in concentrate and as regards grade of concentrate is about what should be expected. Had less dressing water been used per ton fed to the conical deck the copper recovery made by this deck would undoubtedly have been

raised to the level of the recovery made by conoidal deck No. 2, but the grade of the concentrate would have been correspondingly lowered. The gallons of dressing water used per ton fed and some further comparative data are shown by the following figures.

	Conoidal Deck No. 1	Conoidal Deck No. 2	Step Deck	Conical Deck
Gallons of dressing water used	4,780	4,190	15,620	10,210
Tons fed	9.909	9.735	9.699	9.296
Gallons of dressing water used per ton	483	430	1,610	958
Ratio of concentration, tons into one	6.485	2.689	4.334	3.949
Ratio of enrichment (copper)	4.316	2.129	2.942	2.838
Copper in concentrate No. 1, per cent. of total concentrate copper	96.6	94.2	74.8	84.2

It should be noted that the total concentrate produced by conoidal deck No. 1 (chord of arc on 1.25-in. slope) is of a much better grade than the No. 1 concentrate produced by any of the other decks, while the recovery of 66.7 per cent. of the copper made by this deck in total concentrate is greater than the recovery made by the other decks in No. 1 concentrate, with the exception of conoidal deck No. 2, which on account of its flat slope at the head of the deck, makes a 7 per cent. higher recovery in No. 1 concentrate than is made by conoidal deck No. 1 in total concentrate, the concentrate being, of course, of very much lower grade.

The total recovery made by conoidal deck No. 1 and the grade of concentrate produced will now be compared with the recovery in concentrate No. 1 made by conoidal deck No. 2 and the conical deck. As the step deck made a concentrate No. 2 of high enough grade to be considered in the finished-concentrate class, its recovery in total concentrate and grade of concentrate will also be placed in comparison.

	Conoidal Deck No. 1	Step Deck	Conoidal Deck No. 2	Conical Deck
	Total Concentrate		No. 1 Concentrate	
Per cent. of copper fed, recovered	66.67	67.86	74.65	60.64
Assay per cent. copper in concentrate	12.56	6.26	7.26	10.87
Assay per cent. insoluble in concentrate	41.6	69.3	64.9	48.7
Assay per cent. FeO in concentrate	22.0	10.6	12.4	18.8

The No. 2 concentrate or middling made by conoidal deck No. 1 and by the step deck are the only ones which can be considered as concentrate, the concentrate No. 2 from conoidal deck No. 2 being of much lower grade than the feed, and the middling from the conical deck being of about the same grade as the feed.

In May, 1913, conoidal deck No. 3 was installed in the slime plant. The deck was given a cement surface of finish similar to that on the other two conoidal decks installed in the slime plant.

This deck varied from the first two conoidal decks installed in that the chord of the arc was laid on a slope of 1.125 in. per foot instead of 1.25 in. per foot for conoidal deck No. 1, or 1 in. per foot for conoidal deck No. 2. This was the only variable.

The three conoidal decks and the steel-frame conical deck were tested in parallel during the eight-day period May 17 to 24, inclusive, 1913, the results obtained from these tests being presented below. The conoidal-deck results have been placed in the order of decreasing slope of deck surface. This arrangement brings out more clearly the differences in the work of the conoidal decks, and also places the results of conoidal deck No. 2 by the side of those of the conical deck, the work done by these tables being very similar.

Comparative Results on Four Round Tables

	Conoidal			Conical
	No. 1	No. 3	No. 2	
Slope of chord of arc in inches per foot	1.25	1.125	1.00	.....
Feed:				
Gallons of feed pulp per 24 hr	14,546	15,427	15,280	16,341
Tons of dry solids in feed per 24 hr	6.526	6.887	6.882	7.007
Density of feed in grams per gallon	407	405	405	389
Assay per cent. copper in feed	2.67	2.66	2.46	2.37
Pounds of copper fed per 24 hr	349	367	336	332
Concentrate No. 1:				
Gallons per 24 hr	10,522	11,672	10,523	13,562
Tons dry solids per 24 hr	0.821	1.280	1.643	1.489
Density in grams per gallon	71	99	142	100
Per cent. of solids in feed	12.58	18.59	24.08	21.24
Assay per cent. copper	13.03	9.84	7.03	7.56
Assay per cent. insoluble	39.6	53.4	65.5	63.3
Assay per cent. FeO	22.9	17.4	12.3	13.3
Pounds of copper per 24 hr	214	252	231	225
Per cent. of copper in feed	61.32	68.66	68.75	67.77
Concentrate No. 2 (middling):				
Gallons per 24 hr	2,005	2,206	4,107	4,587
Tons dry solids per 24 hr	0.101	0.193	0.442	0.588
Density in grams per gallon	45	79	98	116
Per cent. of solids in feed	1.54	2.80	6.49	8.39
Assay per cent. copper	4.97	2.59	1.81	1.79
Assay per cent. insoluble	74.6	84.7	88.1	88.3
Assay per cent. FeO	8.0	4.1	2.8	2.7
Pounds of copper per 24 hr	10	10	16	21
Per cent. of copper in feed	2.86	2.73	4.76	6.33

	Conoidal			Conical
	No. 1	No. 3	No. 2	
Total concentrate:				
Gallons per 24 hr	12,527	13,878	14,630	18,149
Tons dry solids per 24 hr	0.922	1.473	2.085	2.077
Density in grams per gallon	67	96	129	104
Per cent. of solids in feed	14.12	21.39	30.57	29.63
Assay per cent. copper	12.15	8.89	5.92	5.92
Assay per cent. insoluble	43.4	57.5	70.3	70.4
Assay per cent. FeO	21.3	15.6	10.3	10.4
Pounds of copper per 24 hr	224	262	247	246
Per cent. of copper in feed	64.18	71.39	73.51	74.10
North-side tailing:				
Gallons per 24 hr	8,519	9,718	8,555	9,634
Tons dry solids per 24 hr	2.288	2.602	2.037	2.261
Density in grams per gallon	244	243	216	232
Per cent. of solids in feed	35.05	37.78	29.85	35.12
Assay per cent. copper	0.98	0.96	1.08	0.93
Pounds of copper per 24 hr	45	50	44	46
Per cent. of copper in feed	12.89	13.62	13.10	13.85
South-side tailing:				
Gallons per 24 hr	10,852	9,289	10,593	9,351
Tons dry solids per 24 hr	3.317	2.812	2.700	2.470
Density in grams per gallon	277	275	231	240
Per cent. of solids in feed	50.83	40.83	39.58	35.25
Assay per cent. copper	1.21	0.98	0.83	0.81
Pounds of copper per 24 hr	80	55	45	40
Per cent. of copper in feed	22.93	14.99	13.39	12.05
Total tailing:				
Gallons per 24 hr	19,371	19,007	19,148	18,985
Tons dry solids per 24 hr	5.605	5.414	4.737	4.931
Density in grams per gallon	262	258	224	236
Per cent. of solids in feed	85.88	78.61	69.43	70.37
Assay per cent. copper	1.11	0.97	0.94	0.87
Pounds of copper per 24 hr	125	105	89	86
Per cent. of copper in feed	35.82	28.61	26.49	25.90
Ratio of concentration (No. 1 concentrate)	7.9	5.4	4.2	4.7
Ratio of concentration (total concentrate)	7.1	4.7	3.3	3.4
Ratio of enrichment, copper (No. 1 concentrate)	4.9	3.7	2.8	3.2
Ratio of enrichment, copper (total concentrate).	4.6	3.3	2.4	2.5
Gallons of fresh water used per 24 hr	17,352	17,458	18,498	20,793
Gallons of fresh water used per ton fed	2,659	2,535	2,711	2,967

Discussion of Results of Eight-Day Test

The following tabulation shows that the test results arrange the different decks in just the order which should be expected both in regard to copper recovery and grade of concentrate produced.

Order of Deck Efficiency Based on Total Concentrate

In Regard to Copper Recovery		In Regard to Grade of Concentrate			
	Recovery, Per Cent.		Cu	Insol.	FeO
Conical deck	74.10	Conoidal deck No. 1	12.15	43.14	21.3
Conoidal deck No. 2	73.51	Conoidal deck No. 3	8.89	57.5	15.6
Conoidal deck No. 3	71.39	Conoidal deck No. 2	5.92	70.3	10.3
Conoidal deck No. 1	64.18	Conical deck	5.92	70.4	10.4

The work of the conical deck is practically identical with the work of the conoidal deck No. 2. The conoidal deck results illustrate very clearly the effect of the slope on the work of the table, the grade of concentrate produced being lowered by a decrease in slope while the recovery of copper is raised. Conoidal deck No. 3, with the chord of its arc laid on a slope of 1.125 in. per foot, was designed to produce a cleaner grade of concentrate than the conical deck, with its slope of 1.25 in. per foot, at the same time making practically the same recovery of copper. The tests showed that the results desired were very closely approximated, conoidal deck No. 3 recovering 71.39 per cent. of the copper fed in 21.39 per cent. of the total feed solids while the conical deck recovered 74.10 per cent. of the copper fed in 29.63 per cent. of the feed solids. In considering these results it must be remembered that the conical deck had a radial width 6 in. greater than the radial width of the conoidal deck and that, therefore, in order to make the results strictly comparable, the deposit on the lower 6 in. of the conical deck should have been washed into the tailing launder.

Order of Deck Efficiency Based on Concentrate No. 1

In Regard to Copper Recovery		In Regard to Grade of Concentrate			
	Recovery, Per Cent.		Cu	Insol.	FeO
Conoidal deck No. 2	68.75	Conoidal deck No. 1	13.03	39.6	22.9
Conoidal deck No. 3	68.66	Conoidal deck No. 3	9.84	53.4	17.4
Conical deck	67.77	Conical deck	7.56	63.3	13.3
Conoidal deck No. 1	61.32	Conoidal deck No. 2	7.03	65.5	12.3

The above tabulation shows that the conical deck and the conoidal deck No. 2 did very similar work in producing a finished concentrate,

the conical deck making a slightly cleaner grade of concentrate and a slightly lower recovery of copper.

The tabulation also shows that conoidal deck No. 3 made a higher recovery in a much cleaner grade of finished concentrate than the conical deck, recovering 68.66 per cent. of the copper fed, contained in 18.59 per cent. of the original feed solids, while the conical deck recovered 67.57 per cent. of the copper fed, contained in 21.24 per cent. of the original feed solids.

After making a few tests on the first conoidal deck installed in the slime plant, the writer became impressed with the fact that the conoidal deck surface presented possibilities for producing a cleaner grade of concentrate than is produced by a conical deck with a slope of 1.25 in. per foot while at the same time making as high a recovery of copper, and that a conoidal deck could be installed which would yield just such results as are recorded above for conoidal deck No. 3. The recovery made by a conical deck can, of course, be increased by a slight flattening of the slope, the increase in recovery being made at the cost of lowering the grade of the concentrate, or the grade of concentrate may be enriched by a slight increase of the slope, the enrichment being made at the cost of a lower recovery of copper. Thus, at first glance, it might appear that by merely juggling with deck slopes or feed densities, the feed density being fully as important as the slope, the results obtained upon any conoidal deck could be duplicated upon any conical deck, or *vice versa*. While the writer is not yet prepared to state emphatically that such is not the case, it may be stated that a conoidal deck can be installed which will make a cleaner grade of concentrate and as high a recovery of copper as a conical deck with a slope of 1.25 in. per foot. It may be possible that further experiment may develop the fact that a combination of the two types of deck surfaces will be beneficial, making the deck semi-conoidal, semi-conical, the upper part of the deck being conical.

Qualitative Sampling of Concentrates Across Conical Deck

Early in August, 1912, the writer, at the suggestion of Mr. Goodale, obtained qualitative samples of the rough concentrate on the deck of the steel-frame conical table for each foot of deck surface from the center to the perimeter. The assay results on these samples showed that the concentrate deposit at the head of the deck was considerably richer than the finished concentrate produced by the slime plant, while the deposit nearer the perimeter of the deck was of very low grade. The table was tested on August 11, 12, and 13, being sampled quantitatively preceding the qualitative sampling of the concentrate, two or three revolutions of the table being allowed to elapse before the qualitative samples were taken.

The density of the feed was increased with each day's sampling, the light density of the first day making a lighter tonnage of feed to the table than was the case on the two following days. The quantitative results of the tests will first be presented:

	Date of Test		
	Aug. 11	Aug. 12	Aug. 13
Rate of feed, tons per 24 hr	9.10	11.50	11.05
Density of feed in grams per gallon	386	454	482
Assay, per cent. copper in feed	2.00	2.50	2.44
Tons of concentrate produced per 24 hr	1.687	3.112	4.151
Assay per cent. copper in concentrate	6.05	6.45	4.70
Assay per cent. insoluble in concentrate	71.0	71.6	77.2
Assay per cent. FeO in concentrate	8.7	8.7	7.8
Per cent. of copper fed, recovered in concentrate	56.04	69.86	72.36
Ratio of concentration, tons into one	5.4	3.7	2.7
Assay per cent. copper north tailing	1.05	1.05	1.05
Assay per cent. copper south tailing	1.10	1.00	1.10
Assay per cent. copper total tailing	1.08	1.03	1.08
Gallons of fresh water used per 24 hr	12,803	10,800	12,126
Gallons of fresh water used per ton fed	1,407	939	1,097
Gallons of fresh water used per ton concentrate	7,589	3,470	2,921

On August 11, the table was receiving a low grade of feed and, therefore, making a lower percentage recovery of copper than would be made with a richer feed. The feed was also more dilute than on the two following days, which may have helped the tailing loss to some extent. On August 12, a slightly richer and denser feed was treated and an increase made in the recovery of copper in concentrate, the grade of the concentrate being slightly better as regards copper value than that made on the preceding day.

On August 13, the table feed had about the same assay value as on the preceding day, but had a greater density. The percentage of copper in feed recovered in concentrate showed a further increase, chiefly due to the lower grade of concentrate produced. (In a test made on the above deck in May, 1911, a 73 per cent. recovery of copper was obtained in a concentrate assaying 7.03 per cent. copper, 69.1 per cent. insoluble, and 9.4 per cent. FeO. This recovery was made on a feed assaying 2.91 per cent. copper, and with a rate of feed of 6.2 tons per 24 hr., as against 11.05 tons per 24 hr. in the present test.)

The following tabulation shows the results obtained on the qualitative samples of rough concentrate.

During the test of August 13, the end of the feed box was partitioned off to prevent new feed from coming on the table at the head of the sampling line, therefore, in this case a uniform increase in copper and iron percentages and a uniform decrease in insoluble content of the concentrate was obtained from the perimeter to the feed box. The samples

were taken along a line on the table immediately preceding the wash-off spray, so that the samples may be said to represent the rough concentrate produced by the table.

Analyses of Rough Concentrate from Steel-Frame Conical Table

Sample.	Copper			Insoluble			Ferrous Oxide		
	Aug. 11	Aug. 12	Aug. 13	Aug. 11	Aug. 12	Aug. 13	Aug. 11	Aug. 12	Aug. 13
No.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1.....	3.75	2.25	1.15	80.8	87.2	91.6	4.9	2.9	2.2
2.....	5.5	2.9	2.15	74.0	85.2	87.6	7.5	3.3	3.6
3.....	7.1	3.35	2.2	67.6	83.8	87.4	9.8	4.2	3.2
4.....	8.1	6.95	2.9	63.0	70.6	84.4	11.6	9.1	4.5
5.....	9.35	7.55	4.2	57.6	68.4	79.8	13.9	9.8	6.9
6.....	10.2	12.3	6.0	54.4	48.4	72.8	14.7	17.8	8.6
7.....	9.2[a]	12.1[a]	10.9	57.4[b]	49.4[b]	52.2	13.8[a]	17.4[a]	17.1
8.....	7.35[a]	11.45[a]	13.2	64.8[b]	52.6[b]	40.8	11.3[a]	16.3[a]	22.0
Total Conct.	6.05	6.45	4.7	71.0	71.6	77.2	8.7	8.7	7.8

Sample No. 1 taken from first foot above perimeter of table; sample No. 8 taken from first foot below feed box.

[a] The lowering of the copper and FeO percentages at this point was caused by new feed coming on the table.

[b] The increase in insoluble content at this point was caused by new feed coming on the table.

The results obtained on August 13 are shown graphically in Fig. 3. The curves illustrate the results obtained when the new feed is excluded from the two upper samples.

The results showed that in using the conical round tables as roughing tables, which was the regular slime-plant practice at the time the samples were taken, a grade of concentrate was being produced on the upper part of the deck which was of a better grade than the usual run of finished concentrate, and that in removing this concentrate from the deck it was remixed with a large quantity of very siliceous material from the lower part of the deck.

Another set of qualitative samples was obtained during a two-day test made on August 26 and 28, 1912, the material on the deck surface being sampled just preceding the wash-off spray and also on a radial line diametrically opposite the wash-off spray. In the tabulation (p. 461) "*A*" samples are those taken immediately preceding wash-off spray, and "*B*" samples those taken on a radius diametrically opposite the wash-off spray. Plus and minus signs indicate that results on "*B*" samples are greater or less than results on "*A*" samples.

Another set of qualitative samples was taken on June 17, 1913, including samples taken on each of the three conoidal decks which were in operation at this time and on one of the steel-frame conical decks.

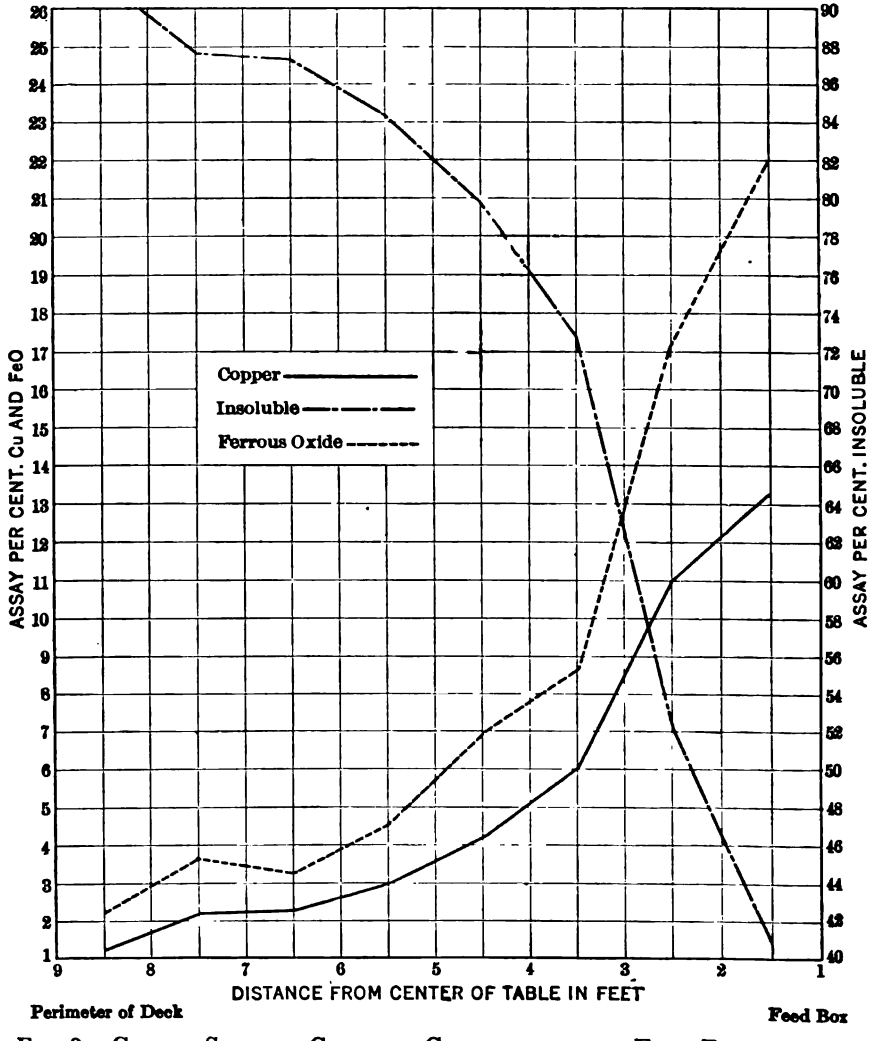


FIG. 3.—CURVES SHOWING GRADE OF CONCENTRATE FOR EACH FOOT OF DECK SURFACE OF STEEL-FRAME CEMENT-DECK ROUND TABLE.

Insoluble curve, one-half scale of Cu and FeO curves.

It was at first intended to make a more extended test in this direction but it was afterward decided that enough experimental data had been secured to answer all practical purposes.

Comparison of Average Results of "A" and "B" Samples

Sample No.	Copper			Insoluble			Ferrous Oxide			Sulphur		
	A Per Cent.	B Per Cent.	Difference Per Cent.	A Per Cent.	B Per Cent.	Difference Per Cent.	A Per Cent.	B Per Cent.	Difference Per Cent.	A Per Cent.	B Per Cent.	Difference Per Cent.
1...	1.80	1.33	−0.47	89.2	90.3	+0.11	2.7	2.2	−0.5	2.8	1.9	−0.9
2...	3.17	2.16	−1.01	83.7	87.7	+4.0	4.6	3.4	−1.2	5.3	3.4	−1.9
3...	3.94	2.99	−0.95	80.5	84.4	+3.9	6.0	4.3	−1.7	6.7	4.8	−1.9
4...	4.37	4.36	−0.01	78.9	78.9	.....	6.5	6.2	−0.3	7.5	7.3	−0.2
5...	6.97	6.72	−0.25	68.2	69.1	+0.9	10.7	10.2	−0.5	12.1	12.0	−0.1
6...	9.60	8.85	−0.75	57.5	60.2	+2.7	14.7	13.6	−1.1	17.0	15.9	−0.1
7...	11.51	10.48	−1.03	48.4	52.2	+3.8	18.4	16.8	−1.6	21.3	19.7	−1.6
8...	11.15	8.85	−2.30	48.3	58.3	+10.0	18.6	15.1	−3.5	21.6	17.2	−4.4

Samples are numbered from the perimeter of the deck to the feed box.

The following results, while not absolutely uniform, serve to illustrate the difference in the work of the tables:

Sample No.	Conoidal Decks									Conical Deck		
	No. 1, slope 1.25 in.			No. 3, slope 1.125 in.			No. 2, slope 1.00 in.			Slope 1.25 in.		
	Assay Per Cent.											
	Cu	Insol.	FeO	Cu	Insol.	FeO	Cu	Insol.	FeO	Cu	Insol.	FeO
1	12.1	44.5	20.0	3.62	80.7	4.9	2.5	85.1	3.9	2.23	86.8	3.1
2	12.1	45.7	19.1	5.65	73.9	7.4	4.6	76.2	7.6	2.05	87.1	2.9
3	12.9	41.8	21.0	6.53	69.7	9.1	4.1	78.5	6.7	2.01	87.6	2.7
4	10.9	50.0	18.0	5.50	73.2	8.6	4.8	74.9	7.9	2.95	83.8	4.4
5	9.1	58.0	14.7	4.9	76.0	7.4	4.7	75.9	7.8	3.56	81.2	5.4
6	9.5	56.6	15.3	5.1	74.4	8.5	4.9	76.7	7.6	6.03	70.0	10.3
7	10.6	50.2	18.1	6.2	69.3	10.6	5.6	71.8	9.7	9.22	54.9	17.0
8	14.4	33.4	25.9	9.4	54.3	17.5	7.4	62.8	13.6	12.59	40.2	23.3

Samples are numbered from perimeter to center of table. At the time the first of above samples was taken, and for some two years previous to this time, all of the slime-plant round tables, with the exception of one table which was used for clearing up finishing-table middling, were operated as roughing tables, the rough concentrate being cleaned on Deister No. 3 slime tables. This roughing system, or two-stage system, was installed under the supervision of G. M. Bates, slime-plant foreman, along the lines recommended by Dr. Richards some time previously, with the exception that the tables were set to make 3 rev. an hour instead of 1 rev.

This system gave very satisfactory results, but, of course, necessitated

the re-treatment of a large amount of material and the continuous circulation of a large quantity of middling.

On a conical deck round table, operating as a roughing table, taking an average of all decks in operation, about 70 per cent. of the original feed copper is concentrated in about 30 per cent. of the original feed solids, and about 30 per cent. of the copper and 70 per cent. of the feed solids are sloughed off as tailing. Hence, under above conditions, about 30 per cent. of the original feed to the round tables requires re-treatment on finishing tables before a satisfactory concentrate can be obtained.

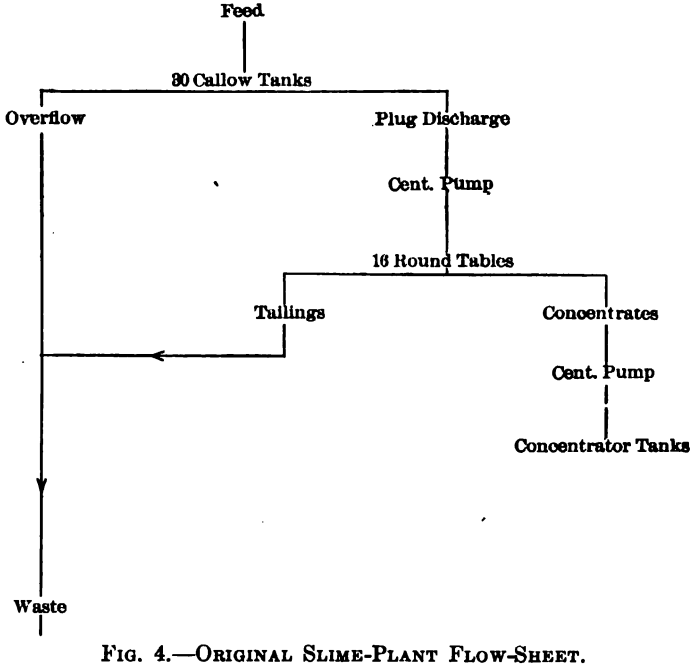


FIG. 4.—ORIGINAL SLIME-PLANT FLOW-SHEET.

After some further investigation of the grade of the rough concentrate on various parts of the deck surface of the round tables, it became evident to the writer that it was unnecessary to re-treat the whole of the rough concentrate, as by far the greater proportion of the concentrate copper was contained in a concentrate which was clean enough for a finished product.

Arrangements were therefore made to operate the round tables in the slime plant to make two products in addition to the tailing product, a finished concentrate and a rough concentrate or middling, the latter product only being subject to re-treatment.

The necessary changes for operating the plant as noted above were completed under the supervision of Mr. Bates and the plant put into operation on Mar. 1, 1913.

Summary of Slime-Plant Practice

The following summary, submitted by George M. Bates, foreman of the slime plant, illustrates the progress in slime-plant practice as the result of the application of the knowledge gained by sampling. Four successive

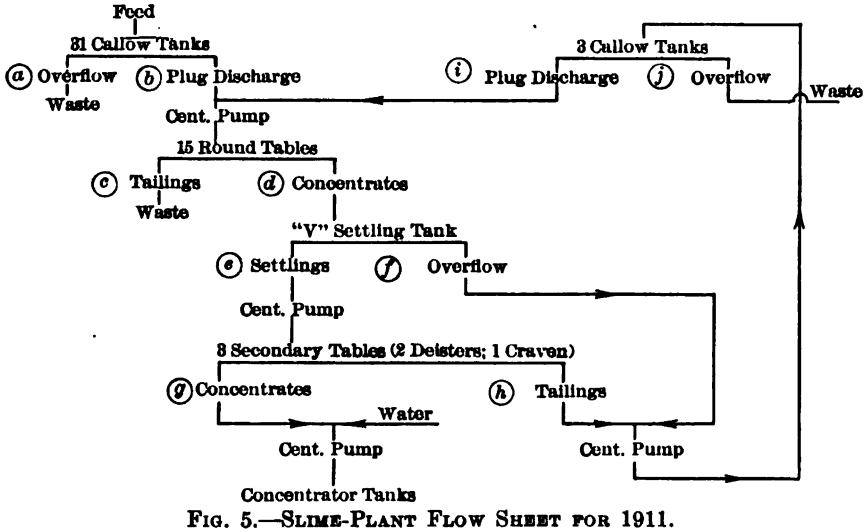


FIG. 5.—SLIME-PLANT FLOW SHEET FOR 1911.

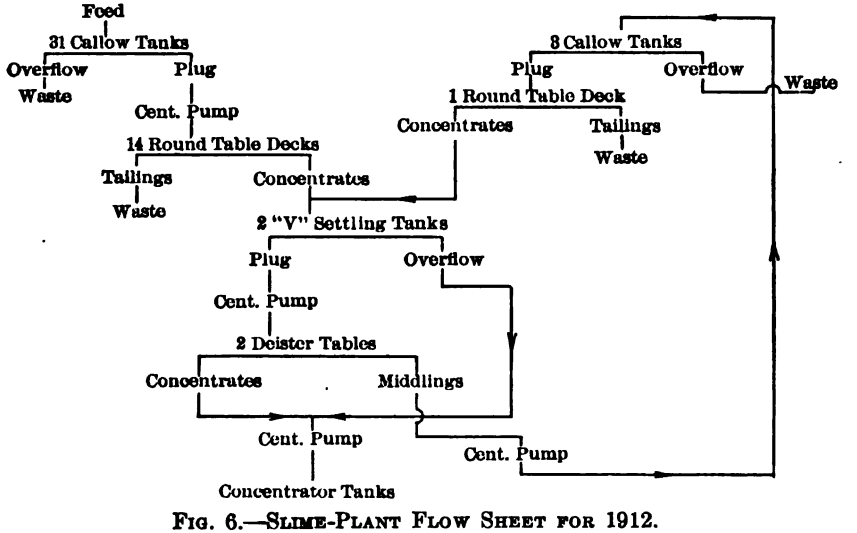


FIG. 6.—SLIME-PLANT FLOW SHEET FOR 1912.

flow sheets are presented, followed by a tabulation of figures to show the improvement at each step forward.

"Fig. 4 illustrates the original slime plant, as operated from 1905 to

1910. Three kinds of deck surface were in use, viz., plain wood, linoleum, and smooth cement. All, however, were operated for two finished products, concentrate and tailing.

"Fig. 5 illustrates the so-called roughing system. The round-table decks were canvas covered and operated primarily for finished tailing. The rough concentrate was re-dressed on Deister tables producing finished concentrate and a middling which was dewatered and returned to the round-table system.

"In this connection the writer wishes to call attention to the fact that this roughing system is more than its name implies. Previous to this time the round tables had probably operated as true film-sizing tables. But, with the adoption of the roughing system, the tables ceased

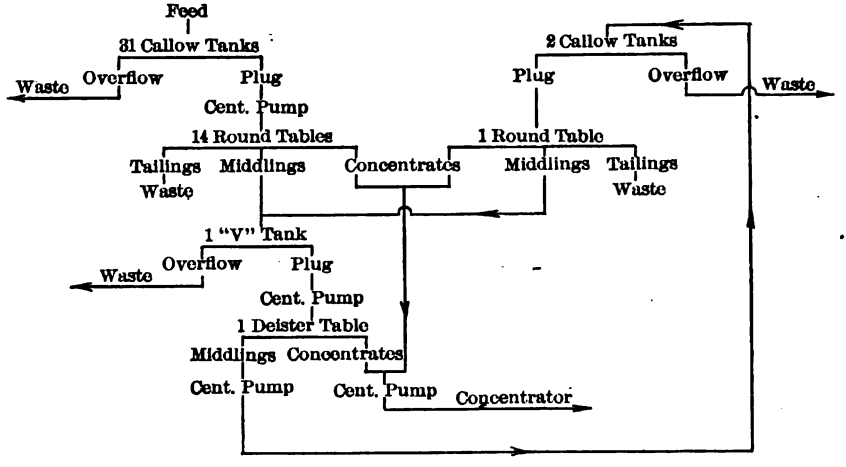


FIG. 7.—SLIME-PLANT FLOW SHEET ADOPTED MARCH, 1913.

to be film sizers, and became building tables. The feed began to bed on the tables, sometimes as much as ⅜ in. in thickness. The rough canvas surface undoubtedly assisted in the rapid formation of this bed. After this bed was once established, however, the character of the deck surface at that point would seem to be of small moment, except in so far as it tended to hold back quartz grains and thus rendered washing more difficult. Whatever enrichment took place in the bed itself, as it was subjected to the action of fresh feed, must have been due to replacement of quartz grains by mineral grains.

"Fig. 6 illustrates the revised roughing system or building-table system, in which the return middling, instead of being returned to the round-table system, was subjected to an individual treatment on a single round table. The equipment at this time included several rough cement decks. The cement surface was adopted, not for any particular virtue in itself, but as an economical substitute for canvas. The cost of mainte-

nance of the canvas surface was high. It was evident that if a cement surface of similar character could be produced it would be much cheaper to maintain. Upon trial it was found that the cement surface had the added advantage of requiring less wash water for the removal of the concentrate.

"Fig. 7 illustrates the latest slime-plant practice and a distinct step forward. The space samples taken at regular intervals from the center to the perimeter of the round table (described by Mr. Crowfoot) showed that the round table would produce a certain amount of high-grade concentrate directly. The operation of the plant was immediately changed to conform to this knowledge. The rough surface deck was retained (either canvas or cement) but the tables were equipped to make three products, viz., a finished tailing, a certain amount of finished concentrate, and a considerable amount of middling, the latter then enriched on a secondary table (Deister or James). The return middling was re-treated on a round table as before. While Mr. Crowfoot's tests indicated that a very high-grade concentrate, namely from 10 to 14 per cent. copper content, was possible under this system, owing to certain considerations of recovery and smelting costs, the production of a concentrate richer than 8 per cent. copper content has not been attempted.

"The following figures, corresponding to the flow sheets just presented, show that the progress in slime concentration has been real."

Average Monthly Figures of Slime Plant

Flow sheet number	1	2	3	4
Year	1908	1911	1912	1913[a]
Average number round tables	15.5	14.0	14.5	14.9
Table feed:				
Tons per month	2,091	2,273	2,474	3,010
Tons per table per 24 hr	4.70	5.68	6.16	7.28
Assay per cent. copper	2.56	2.63	2.63	2.78
Density in grams per gallon	385	372	386	499
Density in per cent. solids	9.5	9.2	9.5	12.0
Plant concentrate:				
Assay per cent. copper	5.14	9.87[b]	9.27[b]	8.02[c]
Copper content, pounds	56,160	64,180	47,420	114,680
Tailing:				
Assay per cent. copper	1.65	1.38	1.33	1.15
Percentage copper recovery:				
Plant	40.4	39.8	43.0	50.7
Tables	52.4	53.6	57.2	68.6

[a] April to September.

[b] Final concentrate after being finished on one or two Deisters.

[c] Final concentrate = round-table concentrate plus secondary table concentrate.

Suggested Flow Sheet for Slime Classification and Treatment

While testing out the conoidal decks in the slime plant the writer took up the question of classifying the through 0.07-mm. slime preceding round-table treatment with a view to obtaining a further increase in slime-plant efficiency, having in mind the following flow sheet for the treatment of slime.

(1) The slime overflowing the hydraulic classifiers in the mill to be sent to V-settling tanks equipped with transverse or longitudinal baffles set at an angle of 45° to insure the overflowing of colloidal or semicolloidal material only, the fine crystalloid material to be discharged with a certain amount of the colloidal material through the spigot.

(2) The spigot discharges of the V-tanks, a comparatively small volume of slime, to be sent to cone-shaped hydraulic classifiers, designed to effect a separation between the colloidal and the crystalline material. The spigot discharge of the cone classifiers to be fed to fine-sand tables of the Wilfley or James type or to round conoidal deck tables having a comparatively flat slope. The overflow of the classifiers to join the overflow of the V-tanks.

(3) The combined overflow product of the V-tanks and cone classifiers to be dewatered in Callow tanks equipped with conical baffles set at an angle of 45° (about seven baffles per tank). Some of the very finest of the colloidal material to be sloughed off as waste in the overflow of the Callow tanks, probably about 10 per cent. of the total solids fed. The overflow of the Callow tanks to go to waste, or to be pumped for re-use in mill; the spigot discharges of the Callow tanks to be sent to free-settling classifiers, or long tanks equipped with longitudinal baffles set at an angle of 45°. The cross-sectional area of the tanks to be large enough to permit the slime to pass through the tanks at a very slow rate. The speed of the current at all points in the cross-section to be kept uniform by the use of a feed sole and the longitudinal baffles spaced 3 in. apart. The overflow of the tanks, if any, to go to waste. The spigot discharge of the tanks to be combined in three products representing head end, center and tail end of tanks. The combinations of the spigot discharges of the tanks to be distributed as follows: Spigots from head end of tank to go to revolving round tables with conical or conoidal deck surfaces and with about the same overall slope as conoidal deck No. 2, previously mentioned (flat slope). Spigots from center of tank to go to similar tables with about the same overall slope as conoidal deck No. 3, previously mentioned (medium slope). Spigots from tail end of tank to go to similar tables with about the same overall slope as conoidal deck No. 1, previously mentioned (steep slope).

All round tables to be operated to produce a finished concentrate, a middling for re-treatment, and a waste tailing product.

The re-treatment middling to be pumped back to head of Callow tank system, to go over again. No finishing tables other than the round tables to be employed.

In connection with the classification of through 0.07-mm. slime, the writer carried out a laboratory test on the classification of slime by free settling, and on the subsequent concentration of the settled products, the results of which will be presented in another paper. The wet mechanical concentration of the through 0.07-mm. slime resulting from the milling of the copper and iron sulphide ores of the Butte district of Montana is still in process of development, although rapid progress has been made in increasing the efficiency of the practice since Mr. Callow introduced the use of the round table as a concentrating machine for the slime in 1904.

Conclusion

As stated earlier in this paper, the round table has been able to maintain its supremacy over all other types of machines on this particular class of work, and in the opinion of the writer it will continue to do so as long as wet mechanical concentration continues to be the most satisfactory method of extracting the valuable minerals from this class of material.

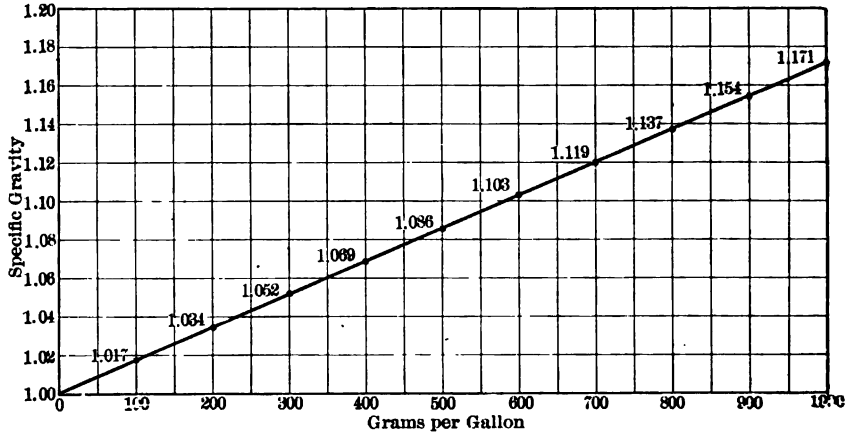
In making the above statement the writer does not mean to convey the impression that the revolving convex round-table practice described in this paper represents the last word in round-table practice on fine slime; it is possible that with further experimenting we may find that decks of a lesser diameter can be used, or, that a combination of the conical and the conoidal deck may prove effective. In the opinion of the writer, the work of the conical steel-frame cement deck would be greatly improved if a smoother finish were given to the deck surface than has so far been used. The comparatively rough cement deck surface used is an excellent surface for saving slime mineral grains but offsets this advantage to a large extent by retaining a large amount of fine gangue material, especially upon the lower radial half of the deck surface, which includes by far the greater part of the deck area. On this area, the transporting power of the water film having been lessened, a considerable amount of fine gangue material settles out which should be kept in a state of semi-suspension if a clean grade of concentrate is to be produced. On the upper radial half of the deck surface a rough finished surface is permissible because a heavy deposition of free-mineral grains occurs on this area which fills up the minute pits in the deck surface and tends to displace any of the lighter grains of gangue material which may settle out on this area.

Therefore, a cement deck surface might be constructed with the upper radial half rough finished and the lower radial half very smoothly finished. It is probable, however, that the result obtained would closely approximate the results obtained from a conoidal deck having the chord of the arc laid on a slope of 1.125 in. per foot and having a rough finished cement surface.

In concluding this paper the writer wishes to acknowledge his indebtedness to the following gentlemen for data furnished, the source of which is not fully acknowledged in the preceding pages of this paper.

Henry Fisher and C. H. Benedict of the Calumet & Hecla Mining Co., John A. Church of New York, Gordon S. Duncan of the Mines Management Co. of New York and London, Eng., H. Foster Bain of the *Mining and Scientific Press*, and A. E. Wiggin, Anaconda. The writer also wishes to acknowledge the courtesy of C. W. Goodale, A. E. Wheeler, and M. W. Krejci of the management of the Anaconda Copper Mining Co., Boston & Montana Reduction Department, in allowing access to the company's files bearing on the subject and to thank Mr. Goodale for valuable suggestions resulting from his reading and criticism of the manuscript.

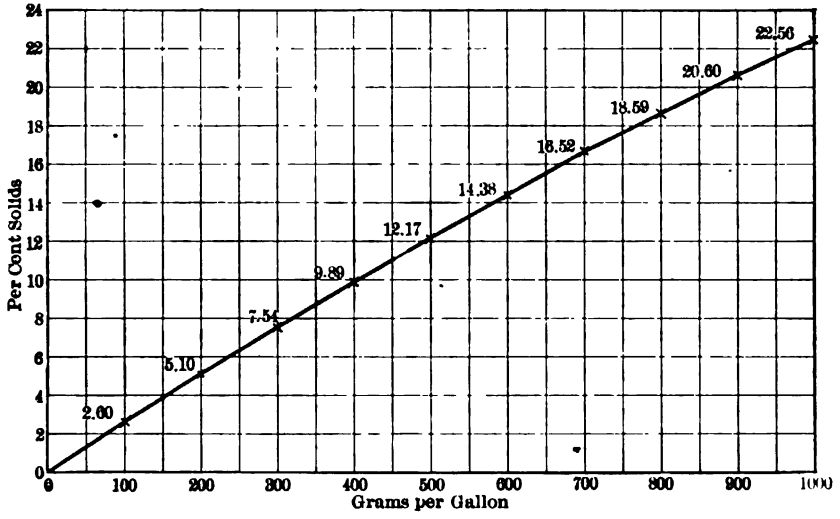
NOTE.—As the expressing of the density of a pulp in grams of solids per gallon of pulp, which is the practice at Great Falls, is probably unique (the usual practice being to express the density in per cent. of solids), three charts, Figs. 8, 9 and 10, are presented with this paper which may be used as follows: Fig. 8: To convert grams per gallon to specific gravity. Fig. 9: To convert grams per gallon to per cent. solids. Fig. 10: To convert grams per gallon to degrees Baumé.



$$\text{Formula: Sp. Gr.} = \frac{3{,}784 - \frac{x}{2.85} + x}{3{,}784}$$

x = Grams per gallon.
3,784 = Weight 1 gal. water in grams.
2.85 = Sp. Gr. of dry slime.

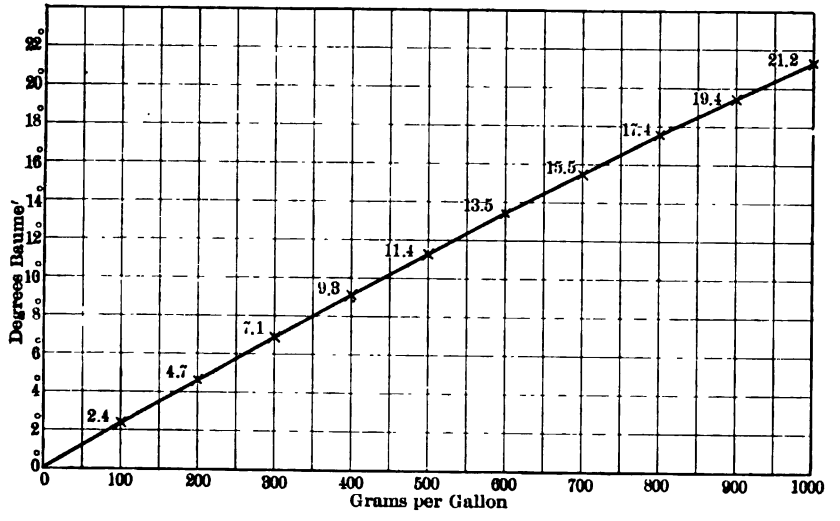
FIG. 8.—SLIME DENSITY. GRAMS PER GALLON TO SPECIFIC GRAVITY.



$$\text{Formula: Per cent. solids.} = \frac{x}{3{,}784 - \frac{x}{2.85} + x}$$

x = Grams per gallon.
3,784 = Weight 1 gal. water in grams
2.85 = Sp. Gr. of dry slime

FIG. 9.—SLIME DENSITY. GRAMS PER GALLON TO PER CENT. SOLIDS.



$$\text{Formula: } B^\circ = 144 - \frac{144}{\text{Sp. Gr.}}$$

FIG. 10.—SLIME DENSITY. GRAMS PER GALLON TO DEGREES BAUMÉ.

The Slime-Concentrating Plant at Anaconda

BY FREDERICK LAIST AND ALBERT E. WIGGIN, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

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I. Introduction

The new slime-concentrating plant at the Washoe Reduction Works, Anaconda, was put into operation during March, 1914. This plant, which has a capacity of 26,000,000 gal. of slime pulp carrying 2,500 tons of solid matter per day, consists of Dorr thickeners and Anaconda multiple-deck slime concentrators. This plant is the culmination of a great deal of experimental work extending over a considerable period of time, which work will be described briefly in the following pages.

II. The Sources and Amount of Slime

Up to the beginning of this year, the ore from the Anaconda company's mines in Butte, Mont., has been concentrated both at the Great Falls

plant and at the Anaconda plant, the former plant treating about 3,000 and the latter about 9,500 tons per day. This year it was decided to do all of the concentrating at the Anaconda plant, which decision of course confines our slime problem to this plant alone. At present larger rolls and additional tables and trommels are being installed in the Anaconda mill to handle the additional tonnage. When milling at the rate of 12,500 tons per day there will be produced 2,500 tons, 20 per cent. of the ore fed to the mill, of material which cannot be concentrated in the mill on account of its extreme fineness. This material is known locally as "slime" and 95 per cent. of it will pass a 200-mesh (0.08 mm.) screen. The flow sheet of the Anaconda mill showing the sources of this slime was given in a paper presented at the Butte meeting, August, 1913, by A. E. Wiggin, entitled The Great Falls System of Concentration installed in Section No. 1 of the Washoe Concentrator at Anaconda.[1]

The 2,500 tons of solid matter is contained in about 26,000,000 gal. of pulp. Heretofore this material has been sent to six large slime ponds 300 by 600 ft. by about 15 ft. deep, which were operated in cycles, three being used in parallel when possible. The settlement was excavated and piled at the end of the ponds by Lidgerwood bucket excavators. This material was then sent to the briquetting plant at the blast-furnace department, where it was used as a binder in briquettes consisting of fine concentrate and fine first-class ore screenings. The smelter could handle not over 25 per cent. of the production of slime and the smelting of this material was expensive on account of its high insoluble content.

III. The Composition of the Slime

Physically the slime may be said to consist of granular and colloidal material. Elaborate settling tests have shown that the colloidal matter amounts to about 35 per cent. of the total material and carries about 21 per cent. of the total copper.

Chemically the slime has the following average analysis:

Cu,	SiO_2,	Al_2O_3,	Sulphur,	CaO,	FeO,	Oz. Per Ton	
Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Ag	Au
2.10	61.0	19.0	4.4	0.6	4.1	1.8	0.005

IV. The Experimental Development of the Anaconda Slime Plant

1. *The Great Falls Slime Plant and Experimental Work*

During 1904, a slime-concentrating plant was erected at Great Falls of sufficient size to treat about one-sixth of the total slime produced in the mill, or about 100 tons of slime per day. This plant consisted of 32

[1] *Trans.*, xlvi, 209 (1913).

8-ft. Callow cones for the thickening of the slime pulp and 16 17-ft. round-table decks for the concentration of the thickened pulp. The round tables were second-hand wooden frame and deck tables which had been displaced in the sand mill by Wilfley tables. The slope of the deck surfaces varied considerably, largely due to the warping of the wooden decks and frames. Later some of the wooden decks were covered with linoleum, rubber, cement, and canvas and these various surfaces tested. It was found that canvas gave the highest recovery but this was at the expense of the grade of the concentrate, while linoleum, rubber, and wood gave a cleaner concentrate, but a lower recovery. Cement seemed to fall between the two groups, although the work of this form of deck is largely dependent upon the character of the finish given the cement. (The "cement" was in reality a concrete containing about one part cement to two parts sand.) It was concluded that the cement surface was the best adapted to the work, particularly because of its durability, and that the framework should be steel, as cement laid over a wooden deck and framing was not at all satisfactory. It was decided then to build a four-deck concrete-and-steel round table, the decks being supported on a central shaft and spaced about 4 ft. apart. Before building this table, however, it was necessary to determine the proper slope of deck surface, as the decks of the tables in the slime plant at that time had slopes varying from 1 to 1½ in. per foot. An extensive series of experiments was carried out during 1909, covering a period of about six months, to determine not only the proper slope of deck surface but also the proper density of pulp, the proper speed of revolution of the table, and the relation between the recovery and the grade of concentrate. A sector of a round table the slope of which could be varied was used in making these experiments. Slopes ranging from ½ in. per foot to 2 in. per foot (4 per cent. to 16 per cent.) were tried, the intervals being ⅛ in. per foot. This work proved conclusively that the best slope of deck surface for the concentration of this slime is 1¼ in. per foot (10 per cent.). At slopes less than 1⅛ in. per foot it was found that a considerable portion of the gangue was deposited beneath the mineral and was not displaced by the mineral; the mineral seemed to form a veritable blanket over the gangue material. This condition, of course, resulted in a very low-grade concentrate as it was impossible to wash out the gangue without a heavy loss of the mineral. At slopes greater than 1⅜ in. per foot the loss of the valuable minerals became excessive, due to the more rapid pulp flow.

It was shown that a density of pulp of from 8 to 12 per cent. solids was best adapted to round-table concentration. Too great a density makes the pulp too viscous and does not permit of a proper separation of the mineral from the gangue material. Too low a density cuts down the capacity of the table unnecessarily. The water contained in the feed pulp plays an important part in the dressing of the concentrate by remov-

ing a great deal of the gangue material. It had been the practice in the Great Falls slime plant to feed the deck over one-half of its surface, dressing with clean water over about seven-sixteenths of the surface and removing the concentrate by strong jets of water on the remaining one-sixteenth of the surface. Our experimental work showed that the deck could be fed to advantage over about three-fourths of its surface leaving about one-fourth for dressing and removing the concentrate. We also determined that the best speed of revolution was from 4 to 6 min. per revolution. This work also showed that it is possible to make a net recovery of 54 per cent. of the copper and silver in a concentrate assaying 6.5 per cent. copper, or better, from a slime feed containing about 2.25 per cent. copper. At the time this work was carried out the slime produced from the Great Falls mill was quite similar in character to that produced in the Anaconda mill to-day. It did settle somewhat more readily, the capacity of settling tanks on the Great Falls slime being about 40 per cent. greater, when settling 95 per cent. of the solid matter, than the same settling tanks when treating the present Anaconda slime.

The Great Falls slime plant as a unit never made a very satisfactory net recovery, due to the fact that the settling capacity was inadequate and to the poor condition of the round tables, which were for the most part second-hand wooden tables. As there was not slime-plant capacity for the treatment of all of the slime produced in the mill it was the policy of the management to overfeed the slime plant and in that way produce more copper, although at a sacrifice in recovery.

A great deal of credit should be given to Dr. R. H. Richards and J. M. Callow for valuable suggestions made during this work at the Great Falls plant.

2. *The Experimental Work Done at Anaconda*

During 1912 the Great Falls system of concentration was installed in the No. 1 section of the Anaconda mill. This system had been developed at Great Falls for the treatment of Butte ore. The round-table concentration of the slime was a part of this system, and in connection with the installation of the system in section No. 1 of the Anaconda mill a slime section consisting of 24 8-ft. Callow settling tanks and two four-deck steel-and-concrete round tables was installed to treat the slime from the remodeled section. The slime section was built to confirm our Great Falls test data and had a capacity of about 425,000 gal. of slime pulp per day, or approximately 20 per cent. of the slime produced in section No. 1 of the mill. We anticipated from our Great Falls data that the 24 Callow tanks would have a capacity of 600,000 gal. when settling at an efficiency of 95 per cent., but, owing to the fact that the Anaconda slime has a lower rate of settling than the Great Falls slime, we found our capacity to be only 425,000 gal. of pulp, or 12.3 gal. per tank per minute, which is equivalent to 0.286 gal. per square foot of settling area

per minute. This amount of pulp carried about 35 tons of solid matter, so that our eight round-table decks were somewhat underloaded.

This experimental slime plant was operated for a period of 14 months during which time a great deal of test work was carried out.

(*a*) *The Slime-Thickening Devices.*—The slime pulp leaves the mill at a density of 2 per cent. solids, and as the mechanical concentration requires a density of about 10 per cent. the first step in the treatment of the slime is to remove the excess water.

Considerable experimental work has been done by C. D. Demond, head of the Washoe Reduction Works testing department, on the use of baffles in tanks for the settlement of slime. We also carried on a series of experiments using the Kuchs-Laist centrifugal colloid separator to eliminate the colloidal material from the slime, treating only the granular portion on the concentrating machines. The work of this machine was very satisfactory and the net recovery from the slime was slightly greater than that made when concentrating the entire slime with the colloid included. It would, however, have taken considerable time to perfect a full-sized machine and for this reason the development of the machine was dropped. A great deal of experimental work was done also upon the Garred filter as a slime-thickening device, but it was found that a filter was not practical for the thickening of such a low-density slime pulp (2 per cent. solids) to a relatively low-density pulp of about 10 per cent. solids. The details of the experiments are given in a paper by Ralph Hayden[2] presented at the Butte meeting of the Institute.

During the early part of 1913, a Dorr continuous thickener, 28 ft. in diameter by 10 ft. deep, was installed at the experimental slime plant. This thickener had a capacity of 195,000 gal. per day when operating at about 95 per cent. efficiency, or 135 gal. per minute, which is equivalent to 0.236 gal. per square foot of settling area per minute. Contrary to our expectations the 28-ft. Dorr thickener showed a slightly less capacity per square foot of settling surface than the 8-ft. Callow cones. The comparative results showed, however, that the settling area is a far more important factor than the length of the overflow weir, as the 24 Callow cones had a total weir length of 560 ft. while the Dorr thickener weir was but 85 ft. long. However, the effective weir length of the Callow tanks was probably not over 280 ft. on account of the very delicate adjustment of the overflow bands that was required when the head on the weir was so extremely small. Realizing the importance of a large settling area we next conceived the idea of a slime-thickening device composed of a number of superimposed shallow trays. Our idea was to feed these trays at one end with the slime pulp, allowing the clear water to overflow at the other end and the thickened pulp to collect in the tray. Periodically the trays would be mechanically tilted at a steep angle and the thickened pulp would be discharged. The trays would then resume

[2] *Trans.*, xlvi, 239 (1913).

a horizontal position and the cycle would repeat itself. We constructed a shallow tray for testing purposes but did not make the tests because this arrangement of trays had suggested to us the possibility of superimposing the Dorr thickener tanks. The Dorr thickeners had the advantage over the tray thickener of continuous operation and we decided, therefore, to drop the tray thickener for the time being. We found that the use of 28-ft. diameter tanks 3 ft. deep permitted of the superimposing of the tanks without getting into too heavy steel work for supports. We therefore made a test to determine the capacity of the 28-ft. diameter Dorr thickener when operated at a depth of 3 ft. This we found to be 160,000 gal. per day, 0.200 gal. per square foot of settling area per minute. This is 85 per cent. of the capacity of the 10 ft. deep tank. At the suggestion of H. N. Spicer of the Dorr Cyanide Machinery Co., we also tested the 28-ft. tank at a depth of 2 ft., and found that at this depth it had about 85 per cent. of the capacity of the 3 ft. deep tank. We finally decided to adopt the Dorr continuous thickener, using tanks 28 ft. in diameter by 3 ft. deep, as being the most economical and efficient form of slime-thickening device. The power required to operate the rakes which scrape the thickened slime to the central discharge is less than $\frac{1}{8}$ h.p. per tank for the 28 by 3 ft. unit. The arms carrying the rakes are set horizontal, or parallel to the bottom, in the 3 ft. deep tank.

The idea of this particular design of Dorr thickener seems to have occurred independently to Mr. Dorr and to ourselves, as so often happens when independent investigators are working along the same lines. Mr. Dorr has since patented this arrangement with our full knowledge.

(*b*) *Slime Concentrators.*—During the first eight months of operation of the experimental plant the four-deck round table was thoroughly tested. These tests included the production of a low-grade concentrate on the round table with subsequent treatment on one of the standard reciprocating-deck slime concentrators. The latter machine made a high-grade concentrate and a rich middling which was returned to the round-table system. It was found that this flow sheet did not give as good commercial results as the production of a clean concentrate directly from the round table requiring no secondary treatment. Among other arrangements we tried treating the round-table middling on a reciprocating table making a concentrate, middling (returned to the round table), and tailing. However, neither this nor the treatment of the round-table middling on a separate round-table deck gave as good results as simply mixing all of the round-table middling with the original table feed for treatment.

A great deal of experimental work was done on the Peck centrifugal slime concentrator, although this work was not carried on at the experimental slime plant.

Several of the standard slime concentrators, which are among the latest types on the market to-day, were tested quite thoroughly. The results of these tests, compared with the work of the round table under

operating conditions, showed that the round table is the most efficient and economical slime concentrator. It was therefore decided to adopt the round table in the new slime-concentrating plant.

As noted previously, the round tables in the experimental slime plant were four-deck machines with the central shaft support and the decks spaced about 4 ft. apart. To make the machine more compact, thereby cutting down the first cost of the plant, we decided to build a 20-deck round table, the decks to be spaced 1 ft. between centers. The last six months of operation of the slime plant was devoted to the development and testing of a 20-deck machine. (See Figs. 3 and 4.)

The entire table was made of steel and concrete. The central shaft support was eliminated, the framework carrying the 20 decks being supported at the periphery by wheels which ran on a circular track. The individual decks consisted of sheet steel, supported by uprights at the periphery but self-supporting at the center, upon which was laid a layer of concrete (1 part cement to 2 parts of sand). This concrete formed the concentrating surface and was given a finish similar to that of a medium-weight canvas. Suitable steel launders were arranged around the outer edge of the decks to receive the products of concentration, two decks discharging into one launder. At the center and extending through each deck was an opening 4 ft. in diameter in which was placed a stationary steel tower. This tower carried the water and feed pulp pipes and a ladderway making each deck easily accessible. The feed pulp and wash water were supplied from a circular launder at the head of each deck. The entire structure carrying the decks was revolved at the rate of 15 times an hour, the power being supplied from a 5-h.p. motor placed beneath the table and engaging through a train of gears a circular rack placed near the periphery of the framework beneath the bottom deck. The actual power required to drive the table under working conditions was 3 h.p. The decks sloped 1½ in. per foot and were 17 ft. in diameter.

The table was operated for about six months and proved to be entirely successful. It was therefore decided to adopt this type of concentrator in the new slime plant. Only a few minor changes were made in the design of the 20-deck table, such as increasing the diameter of the decks to 19 ft. and substituting manganese-steel wheels and rail in place of chilled cast iron. The table occupies a floor area equivalent to a circle about 21 ft. in diameter and the 20-deck table has a height over all, including the automatic feed pulp distributor, of about 35 ft. The 20-deck machine will handle 140 tons of Anaconda slime (dry) per day including the treatment of the middling. The net concentrating area per ton of total feed (including the returned middling) is 37 sq. ft.

The results shown in the following tabulation are typical of the round-table work and are obtained from an eight-day test made during April, 1913. As we did not have sufficient settling capacity at this time to supply feed to the 20 decks only eight decks were used.

Test on 20-Deck Round Table. Middling Returned to Table for Treatment

Product	Rate per 24 Hr.		Assay		Per Cent. of Total	
	Gallons Pulp	Pounds Solids	Cu	Insol.	Solids	Copper
Feed	87,290	93,730	1.95	.....	104.6	102.8
Concentrate	69,000	13,050	7.02	58.8	14.6	51.6
Tailing	134,580	76,540	1.13	.....	85.4	48.4
Total Produced	203,580	89,590	1.98	.....	100.0	100.0

Per cent. of copper in original slime recovered..... 51.6
Tons treated per deck, excluding middling..... 5.6

The round table requires about 3 gal. of dressing water per deck per minute and about 6 gal. of water to remove the concentrate per deck per minute. Fully 80 per cent. of the latter water is recovered in the dewatering of the concentrate and can be re-used.

It is our practice to feed the round-table deck over about 70 per cent. of its surface, supply dressing water over about 18 per cent. of the surface, and to use the remaining 12 per cent. of the surface for the removal of the concentrate. The amount of deck surface required for the dressing water is directly proportional to the grade of concentrate desired, the higher the grade the greater the dressing surface required. Approximately one-half of the material removed by the dressing water enters the middling and is returned to the table for treatment with the original feed. The middling amounts to from 8 to 10 per cent. of the original feed tonnage and carries about 5 to 6 per cent. of the total copper.

(*c*) *Dewatering of the Slime Concentrate.*—The slime-plant concentrate leaves the tables at a density of about 3 per cent. solids. It is of course necessary to dewater this material for treatment in the smelter. We decided to try a Dorr thickener followed by an Oliver filter as a dewatering system. In our experimental work we used a Dorr thickener 28 ft. in diameter by 10 ft. deep, and a 6-ft. continuous Oliver filter. The pulp was thickened to a density of approximately 50 per cent. solids in the Dorr thickener and then fed to the Oliver filter. Our tests showed that the Dorr thickener would treat 250,000 gal. of concentrate pulp containing about 3 per cent. solids and thicken it to a density of from 50 to 60 per cent. solids at practically 100 per cent. efficiency. The Oliver filter had a capacity of about 125 tons of solids contained in a pulp having a density of 55 per cent., when producing a cake carrying not over 15 per cent. moisture. The filter was operated under the "wet" system; that is, one Roots rotary pump handled both air and water in producing the vacuum. The average vacuum maintained was about 16 in.

(*d*) *The Use of Chemicals to Accelerate the Settlement of Slime.*—Ferrous Sulphate.—Two 8-ft. Callow cones were used in a test to determine the efficiency of ferrous sulphate as an accelerator in slime settlement. The two cones were operated under the same conditions of feed, overflow, and spigot discharge, the latter product being maintained at a density of about 10 per cent. solids. To the feed to one cone was added ferrous sulphate in the proportion of 1 lb., 2 lb., and 5 lb. per ton of solid slime. It was found that the addition of 1 lb. per ton gave an increase in capacity of 30 per cent., 2 lb. an increase of 40 per cent., and 5 lb. an increase of 50 per cent.

Salt: Some laboratory experiments were made to determine the effect of salt upon the rate of settlement of slime. Using ¼, ½ and 1 lb. of salt per ton of dry solids caused the slime to settle in only from 3 to 4 per cent. less time than the untreated slime. The salt had no effect upon the settlement of the round-table concentrate.

Glue and Ferrous Sulphate: Tests to determine the efficiency of glue and ferrous sulphate for the settlement of slime were made in a 28 by 10 ft. Dorr thickener. The addition of a mixture of ¼ lb. of each chemical per ton of dry slime increased the capacity of the Dorr thickener from 195,000 to 311,000 gal. of pulp per day, or an increase of 60 per cent. The use of glue and soluble sulphates to increase the rate of settlement of slime is covered by U. S. patent 1065878, held by A. G. French of British Columbia. Some laboratory experiments were made on the use of a mixture of glue and ferrous sulphate to accelerate the settlement of round-table concentrate. It was found that the addition of a mixture of the two chemicals in the proportion of ¼ lb. of each per ton of solids caused the same degree of settlement to take place in 1¼ min. as in 11 min. with the untreated concentrate.

It was decided, however, to install sufficient tank capacity in the new slime plant to thicken the slime at 95 per cent. or better efficiency without the use of any chemical. This of course increased the cost of installation but reduced the subsequent operating expense. However, the use of chemicals offers a cheap and simple means for increasing the settling capacity of a given installation.

(*e*) *The Effect of Temperature upon the Settlement of Slime.*—The range in temperature of the average pulp from our mill is from a minimum of about 38° F. during the winter months to a maximum of 50° F. during the summer months. There is an average daily range of about 6 per cent.

C. D. Demond, head of the testing department of the Washoe Reduction Works, made an interesting series of experiments to determine the effect of temperature upon the rate of settlement of our slime. Portions of slime pulp taken from the main flume during the winter months were used in these experiments. The rates of settlement at the flume

temperature and at other higher temperatures were determined, the pulp being treated by immersing the container in warm water. A depth of pulp of 17½ in. was used and the slime was considered to have settled when the thickened pulp occupied a volume equal to one-fifth of the original. Thus the density had been increased from 2 to 10 per cent. solids. The following table shows the length of time required to settle the Anaconda concentator slime from a depth of 17½ in. to 3½ in.:

Temperature, Degrees F.	Time in Minutes		
	Average	Maximum	Minimum
35	41.7	57.0	29.5
40	38.0	50.7	27.3
45	35.0	45.2	25.0
50	31.7	40.0	23.0
55	28.5	36.0	20.5
60	26.0	33.0	18.3
65	23.3	29.8	16.0
70	21.2	26.7	13.8
75	19.5	25.0	12.0

The tabulation above shows that the *average* slime settles 24 per cent. faster at the maximum summer temperature of 50° than at the minimum winter temperature of 38°. However, we find that the deviations in time from that of the average slime are so great, due to physical differences in the pulp other than temperature, that we may have a slime pulp from our mill which at a temperature of 40° will settle in 27.3 min. while pulp from the same source at another time may require 33 min. to settle at a temperature of 60°, or 20° higher.

Thus it would seem that from a practical standpoint and in designing a slime-settling plant the temperature of the pulp does not play as important a part as other physical characteristics. Undoubtedly the varying ratio of sand to colloid is the most important factor to be considered.

As far as the effect of temperatures met with in ordinary practice upon the recovery of valuable minerals from the thickened pulp on any form of mechanical concentrator is concerned, the operation of our experimental plant covering both summer and winter months showed this to be negligible.

V. Description of the Anaconda Slime Plant

Following is given a brief description of the new Anaconda slime plant which was put into operation on Mar. 13, 1914. The plant consists of three divisions, the slime-thickening division, the concentrating division, and the concentrate-dewatering division. It is designed to treat daily

26,000,000 gal. of slime pulp carrying 2,500 tons of solid matter, this being the amount of slime pulp resulting from the treatment of 12,500 tons of ore in the mill.

1. *Diagrammatic Flow Sheet of Plant*

In Fig. 1 is given a diagrammatic flow sheet of the plant showing number of machines, tonnages of solids per 24 hr., and gallons of pulp per 24 hr.

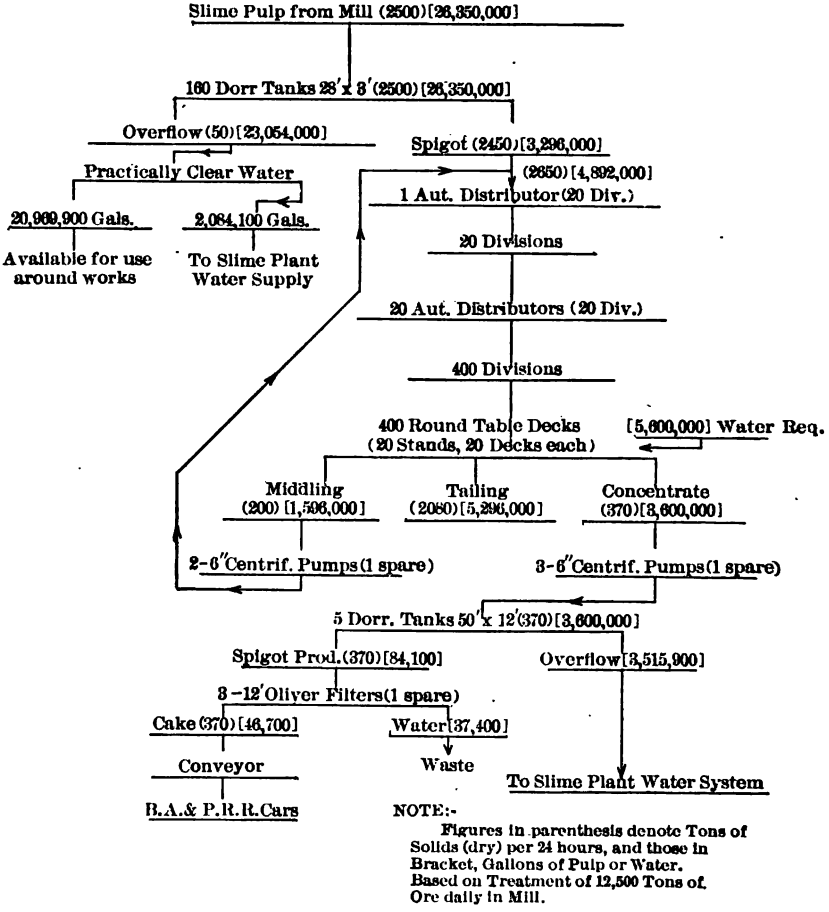


FIG. 1.—FLOW SHEET OF ANACONDA SLIME PLANT.

2. *The Slime-Thickening Division*

The feed to this division consists of the original slime pulp from the mill at a density of about 2 per cent. solids and amounting to about 26,000,000 gal. per day carrying 2,500 tons of solids. This division contains 160 Dorr thickeners 28 ft. in diameter and 3 ft. deep, with central

feed and peripheral overflow, arranged in batteries four tanks high (Fig. 2). Each tank is provided with two rake arms set horizontal and making 1 rev. in 12 min. About ⅛ h.p. is required per tank, or 25 h.p. for the entire division, including transmission. The tanks are operated at an efficiency of about 98 per cent. The overflow water, amounting to about 23,000,000 gal. per day, is available for use in the concentrating division and around the reduction works. The concentrating division requires only about 2,000,000 gal. of this water. The spigot discharge of the tanks,

FIG. 2.—28 BY 3 FT. DORR THICKENERS IN SLIME-PULP THICKENER DIVISION.

which is the feed to the concentrating division, averages about 15 per cent. density. Four men per shift are employed in this division.

3. *The Concentrating Division*

This division receives as feed the thickened pulp from the thickening division, amounting to about 3,333,000 gal. daily carrying 2,450 tons of solids. The pulp passes to a central automatic distributor which divides it into as many portions as there are concentrators in operation. The division contains 20 20-deck Anaconda multiple concentrators (round tables) constructed entirely of concrete and steel. This table is described

in detail earlier in this paper under the heading "Slime Concentrators." Each concentrator is provided with an automatic pulp distributor which gives to each deck its proportionate amount of feed. Under full feed conditions each deck will receive 8,200 gal. of pulp daily carrying 6.1 tons of solids. The middling, amounting to about 1,600,000 gal. daily carrying about 200 tons of solids, or 8 per cent. of the original solids, is returned to the original feed by means of a Traylor 6-in. sand pump. The lift is approximately 45 ft., exclusive of friction. In addition to a middling product the tables make a finished concentrate and a tailing. The concentrate, amounting to about 3,600,000 gal. per day carrying

FIG. 3.—TOP DECK OF ANACONDA MULTIPLE-DECK CONCENTRATOR AND FEED DISTRIBUTOR.

about 370 tons of solids, or 15 per cent. of the original slime, is pumped by two 6-in. Traylor two-stage pumps to the concentrate-dewatering division. The lift, exclusive of friction, is about 105 ft. The tailing is being settled and recovered in the old slime ponds for future treatment. The total concentrating area is approximately 108,000 sq. ft., or 44 sq. ft. per ton of original slime. This enormous concentrating area is contained in a building 275 by 60 ft. Each table requires not over 3 h.p., or about 70 h.p. for the plant, including the driving of the distributors. There are five tablemen, each man having 80 decks under his care, and one pump-

man, employed in this division per shift. The middling and concentrate pumps require about 260 h.p., of which the two concentrate pumps use about 210 h.p.

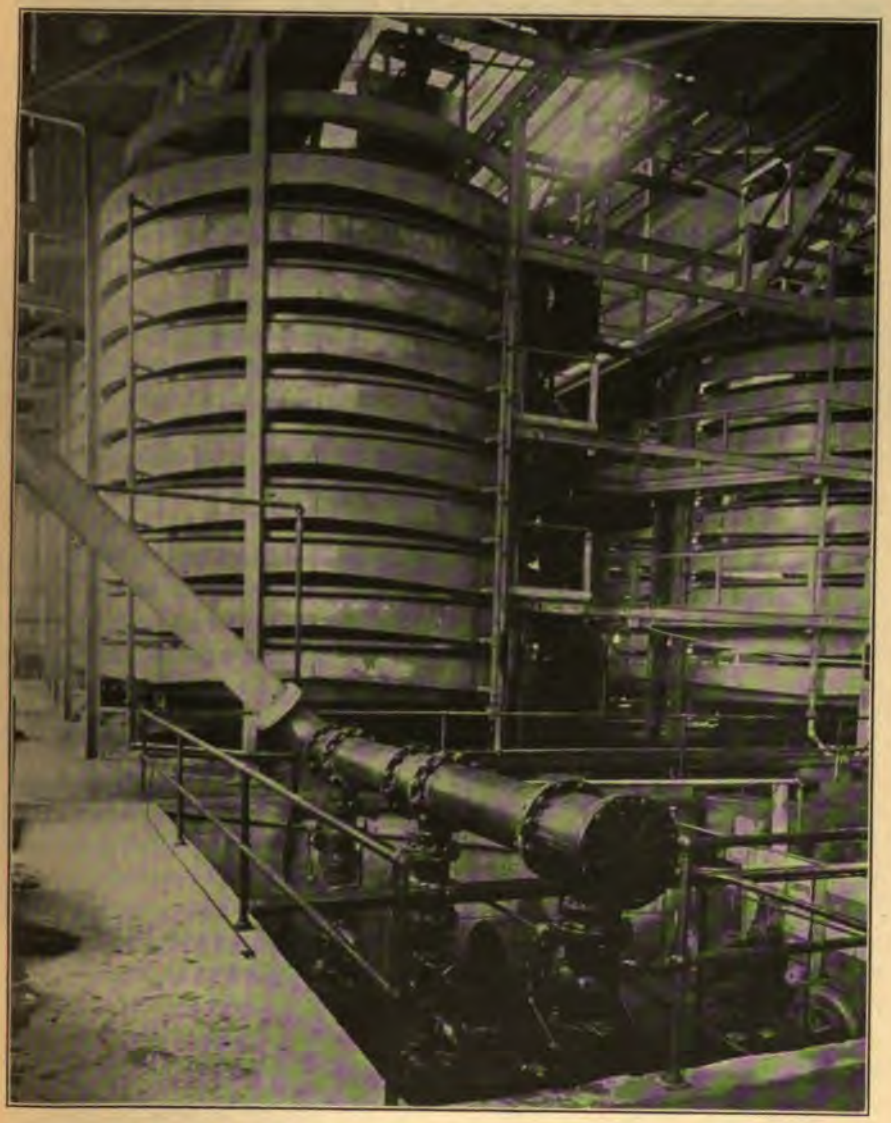


FIG. 4.—ANACONDA MULTIPLE-DECK SLIME CONCENTRATOR.

4. *The Concentrate-Dewatering Division*

The feed to this division—concentrate pulp from the concentrating division—passes first into five 50 by 12 ft. Dorr thickeners. Each thick-

ener is equipped with four rake arms and requires about 1 h.p. The pulp is here thickened from a density of about 2.5 per cent. to one of about 50 per cent. solids. The overflow is clear water and is returned to the water-supply tank for the concentrating division. The thickened pulp passes to two 12 by 12 ft. Oliver continuous filters which produce a cake carrying about 15 per cent. moisture. This is comparatively a dry cake, the material being dry enough to fall apart after pressing in the hand. The cake is discharged on to a conveyor-belt system which dumps directly into 50-ton steel cars for shipment to the smelter. Three men per shift are employed in this division. The Oliver filters require about 30 h.p. each.

5. *General*

In addition to the labor enumerated in the preceding pages there are employed one superintendent on day shift only, one foreman on each shift, one sampler on each shift, and a repairman and helper on day shift only. This makes the total labor requirement 48 men per day. The total power requirement for the plant is about 450 h.p., of which 210 h.p. is required for pumping the concentrate. The total cost of treatment per ton of slime will be about 12c.

The results of the first few weeks of operation have come entirely up to our expectations as regards recovery and grade of concentrate. The plant has operated very smoothly from the start, not having been shut down since it was first put into operation. The entire plant is as nearly automatic in its operation as it is possible to make a concentrating plant.

It is an interesting fact from the economic standpoint that the slime plant is returning fully 200 per cent. on the investment.

The mechanical construction of the plant was carried on in a most satisfactory and efficient manner by the Washoe Reduction Works engineering department, under the direction of U. A. Garred, chief engineer, who was later succeeded by W. N. Tanner. The plant was completed and in operation fully a month and a half before the time set at the beginning of the construction work. The work was under the immediate supervision of W. C. Capron, assistant chief engineer, and George E. Tryon, construction engineer.

A Comparison of the Huntington-Heberlein and Dwight-Lloyd Processes

BY W. W. NORTON, MURRAY, UTAH

(Salt Lake Meeting, August, 1914)

THE gradually increasing proportion of sulphide ores which lead smelters of to-day are called upon to handle has caused the roasting problem to become one of ever greater importance. Mines have increased in depth, methods of concentration have improved, competition in the purchase of ores has become more keen and it has been clear that new and modern methods of eliminating sulphur must receive a large share of attention.

We may look back a score of years or so and recall the old hand roasters, some of which turned out a finely divided roasted product, and others fashioned with a "fuse-box" wherein the roasted ore was slagged. Then came mechanical furnaces of several types, the Brückner cylinder, the Brown-O'Harra, the Ropp, and others, all designed to cut out the cumbersome hand labor of the old reverberatories. Roasting costs were thereupon reduced and tonnages stimulated to a gratifying extent. However, it seems to have been early recognized that this very feature of a greater amount of roasted sulphide ore as compared with oxide ore brought with it a train of difficulties at the blast furnaces. The mechanically roasted ore was fine physically, the blast furnaces were choked and the troubles of the smelterman were not at an end. Then the Huntington-Heberlein pot system of roasting appeared upon the scene, and later on the Dwight-Lloyd machines were invented, and inasmuch as both of these processes were designed not only to roast the ore but to agglomerate it as well, it was apparent that distinct steps in advance had been achieved.

At the Murray plant, modern up-to-date roasting practice is fully exemplified and there are now in successful operation roasting furnaces or devices of several sorts: namely, Godfrey revolving-hearth furnaces, Wedge multiple-hearth mechanical roasters, Dwight-Lloyd sintering machines, and Huntington-Heberlein pots. Godfrey and Wedge furnaces will properly handle material high in sulphur, say ores with 25, 30, and 35 per cent. of that element; D. & L. machines and H. & H. pots will positively not treat efficiently ores or mixtures containing anywhere near the sulphur content mentioned, but are confined to charges containing in the neighborhood of 15 or 18 per cent. In passing, it may also be explained that, so far as the knowledge of the writer goes, Godfrey and Wedge furnaces do not economically eliminate sulphur to an extent

sufficiently low for lead blast-furnace practice. With these simple facts in mind, it will be perfectly clear to all that the metallurgist in charge may elect to treat sulphide ores in either one of two ways: he may pre-roast in Godfrey and Wedge furnaces and subject the partly roasted product to a final treatment on D. & L. machines and H. & H. pots, or he may dilute the average sulphur in the raw ore to 15 or 18 per cent. by means of an admixture of the requisite quantity of non-sulphur fines and send the mixture thus obtained directly to the D. & L. and H. & H. It is believed that there are lead-smelting plants in this country which adopt a straight system of either pre-roasting or diluting, but inasmuch as we seek to discuss Murray-plant methods only it will be sufficient to say that Murray does both. A certain flexibility is thus afforded for a segregation of the various classes of sulphide ores; moreover, in the matter of oxide fines, one can limit screening operations to a point deemed best metallurgically. Unlimited quantities of oxide fines containing any material quantity of lead should not be subjected to the oxidizing action of the final roasters above mentioned.

Godfrey and Wedge furnaces are essentially pre-roasters, D. & L. machines and H. & H. pots are final roasters. At Murray all final roast is either D. & L. or H. & H. We turn now to the primary object of this paper, namely, a brief discussion of the comparative merits of these two widely distinct methods of final roasting.

Cost of Installation

The Murray plant is equipped with two D. & L. machines, the total daily capacity of which may be stated at 220 tons, and 23 H. & H. pots, with capacity of 400 tons. It would, of course, be manifestly unfair to directly compare the total costs of these two installations, but it seems quite safe to say that for almost any given tonnage capacity a D. & L. plant can be built for considerably less than an H. & H. plant, it being understood that by H. & H. is meant the converting-pot portion of an installation only, with no reference to Godfrey furnaces. In the case of the H. & H. one must have heavy cast-iron pots for handling ore in comparatively large units, expensive overhead traveling crane, substantial cooling floor, and, finally, a crusher which the D. & L. does not require at all. The cost of the installation item must be put down in favor of the D. & L.

Cost of Roasting

Any discussion of roasting costs should of course be based on units of sulphur eliminated. In a general way, our experience has shown that the D. & L. will reduce an initial sulphur of about 15 or 16 per cent. to about 4 per cent. in the roasted product, while the H. & H. is capable of handling

a slightly higher initial sulphur, say 17 to 18 per cent., with resultant 5 per cent. in product. During a very recent period of 47 consecutive days, it is known that units of sulphur eliminated per ton of charge by the D. & L. practically equaled units of sulphur eliminated per ton of H. & H., and it is probable that an exhaustive examination of Murray-plant roasting records would show about the same amount of sulphur per ton of charge driven off as between the two sorts of roasters now under review. It follows that figures representing costs of roasting are truly comparable.

The limitations of this paper will not permit of a detailed review of roasting costs, but it may be stated that during the entire year 1913 the H. & H. made the better showing to the extent of about 5c. per ton roasted, and for the first three months of 1914 the H. & H. also had an advantage of about 3c. per ton. Murray experience, everything considered, indicates slightly lower costs for H. & H. as compared with D. & L., but the fact that all calculations are based on operations at an H. & H. plant having twice the capacity of a D. & L. plant must not be overlooked.

Adaptability to Wide Range of Charge Ingredients

Any intelligent discussion of analysis of raw charge to roasters should have the fundamental thought in mind that the metallurgist must treat what comes to the plant. He cannot always be favored with the proportions of silica, iron, and lead which would give the best results, consequently the adaptability of any given roasting device to a variety of materials will be accepted as an item of far-reaching importance.

Some two or three years ago, in connection with a visit to three or four custom lead smelters newly equipped with D. & L. machines, the writer was somewhat impressed with the limitations placed on the charge the machines were capable of handling. Inquiry brought forth the information that certain sorts of materials could be attempted only by resort to a special layer of fine limestone or other infusible material carried next to the grates; any percentage of raw matte at all seemed out of the question; zinc was naturally side-stepped as highly deleterious; much stress was placed upon the proportion of silica to the iron, and nearly all the enthusiasts demanded a goodly percentage of lead provided a choice quality of sinter was to be in evidence. Of late, however, the staff at Murray have demonstrated that a wide range of mixtures may be efficiently handled over the D. & L., and have no doubt that equally good progress has been accomplished at other works. Pre-roasted ore, any kind of raw sulphide ore or concentrates, flue dust, pre-roasted matte, or even raw matte may be combined in certain proportions and successfully sintered over these machines. A sufficient quantity of non-sulphur diluent to bring the average of the mixture down to 16 per cent. sulphur must always be added and of course the details of operation must be cared for.

However, if careful attention to a proper combination of materials at hand into mixtures and close application to operating details have brought results at the D. & L. machines, the same factors are responsible for pleasing results with H. & H. pots. With reference to chemical make-up of charge, the H. & H. will also handle "any old thing."

Turning now to physical character of the raw ore, it is of course recognized that the air currents are required to permeate a thin layer of charge in case of D. & L. treatment, whereas the pot roasters are committed to a very much thicker layer; however, a physically fine charge will restrict tonnage on D. & L. just as surely as it will on H. & H. pots, although the D. & L. process is able to successfully treat slimes or other fine material which it would be wholly useless to attempt to treat on the H. & H.

By way of summing up, it may be stated that the D. & L. process possesses a slight advantage over the H. & H. in the matter of flexibility or range of charge because the D. & L. permits more delicate application of operating details which are essential to success, and also because extremely fine materials find no proper place in the H. & H. charge.

Lead Losses

We have certain data at hand showing a very moderate lead loss on D. & L. machines, these data being based on standard operating conditions during which the resultant gases and fumes were sampled and analyzed. No data are available as covering losses with H. & H. pots. The expense and difficulties in connection with accurately sampling an H. & H. output of 400 tons per day need not be pointed out and gas measurements and samples taken from the combined gases of 23 pots on two different main flues might eventuate in metal-recovery data not wholly dependable.

At the Murray plant there have been periods of time when either the D. & L. or the H. & H. pots have been wholly or partly out of commission, and the metal losses of the whole smelter during these periods have been observed. It is regarded as doubtful if the D. & L. process is productive of any lower metal losses than is the H. & H.

Physical Condition of Product

Final roasting treatment results in a sintered or agglomerated product, and incidentally material of a desirable physical character is passed along to the blast furnaces. The D. & L. sinter is usually of a porous or cellular structure; the H. & H. tends to greater density or firmness. Published and unpublished opinions of metallurgists have sought to show that the peculiarly open or coke-like structure of the D. & L. sinter carried

with it certain extraordinarily favorable properties when subjected to the smelting process in the blast furnace, and have even claimed appreciable saving in the coke percentage used for smelting. Rather exaggerated ideas concerning the efficiency of an exposure of porous surfaces to contact with reducing gases have been advanced and intimate mixtures (possibly intimately combined silica and lead) have been proclaimed as "predigested" and therefore more easily reduced. The writer believes that a partly fused or "predigested" combination may tend to poor results rather than to good results when smelted, for the reason that such substances fuse at too low a temperature in the furnace. Certain writers have gone so far as to examine the cell structures of the D. & L. product microscopically and have declared that glazed or unglazed surfaces have a bearing upon the readiness with which the products were later reduced in furnaces.

With all due respect to the theories above set forth, it was considered that more dependable conclusions could be drawn by means of actual operating tests and accordingly the Murray plant blast furnaces during five days of August, 1912, were run on two charges, the one containing no D. & L. roast at all, the other containing a rather large amount of this material. It was believed that any peculiar virtue existing in D. & L. product would have abundant opportunity to make itself manifest. The exact charges used are given below, together with the average lead in resultant slag and matte.

	Furnaces 1, 3, 7 and 8 (No D. & L.)	Furnace 5 (D. & L.)
Coke, 920 (11½ per cent.)	.....	.....
Bed 36, bin 7	2,970	320
H. & H. roast	2,000	.....
D. & L. roast	.....	4,800
Hand-roasted matte	600	400
Iron ore	690	540
Limestone	1,640	1,840
Scrap iron	100	100
	8,000	8,000
	Per Cent.	Per Cent.
Average lead in slag for the run	0.63	0.91
Average lead in matte for the run	10.7	14.96

Great pains were used to make the experimental run one of value. The D. & L. roasted product was of a typically honeycombed character. No. 5 furnace was in excellent condition, its operations were closely watched by the metallurgist in charge of the furnaces and by the writer, yet absolutely no strengthening of reduction appeared. On the contrary, No. 5 did worse than the other furnaces.

General blast-furnace experience covering a wide range of charges and a considerable period of time indicates that no particular effect, either good or bad, can be claimed for D. & L. sinter as relating to strength of reduction during the smelting process, and exactly the same remark will apply to H. & H. agglomerated material. (Of course the D. & L. sintered cakes must be broken to the proper size and the H. & H. material must be crushed suitably small, or distinctly bad reduction will ensue.)

Leaving now the subject of physical nature of product as affecting reduction we may turn for a moment to physical make-up as relating to speed of blast furnaces. Either of the products of modern roasting development under review is known to help tonnage at blast furnaces enormously. The final roasters of present-day practice have taken the place of the old hand reverberatories and fine-ore-producing mechanical furnaces. A better showing at blast furnaces is the natural result.

As to which product is the better physically, that is to say, which will produce the heavier tonnage at blast furnaces, a first-class D. & L. sinter does not excel a first-class H. & H. agglomerated product. Moreover, given a proposition of inferior quality of both, it would seem that the admittedly cellular or at times fragile D. & L. can hardly equal the more firm and stable H. & H. Here again, however, real experience at blast furnaces may outweigh mere conjecture or theorizing, so the following data are submitted with the idea of showing that in this instance at least the physical character of the D. & L. produced no better tonnage at blast furnaces than did the physical character of the H. & H.

On Aug. 12 and 13, 1912, the following two charges were smelted side by side with the same coke percentage, the same blast pressure, and as near like conditions in other respects as it was possible to obtain.

	Furnaces 1, 3 and 5 (H. & H.)	Furnaces 7 and 8 (D. & L.)
Coke, 920 (11½ per cent.)	.....	.....
Bed 37, bin 4	1,400	2,060
H. & H. roast	3,000	.....
D. & L. roast	.....	3,000
Hand-roasted matte	400	400
Iron ore	1,140	580
Limestone	1,960	1,860
Scrap iron	100	100
	8,000	8,000
Average tons per furnace per day	294	287
	Per Cent.	Per Cent.
Average lead in slag	0.81	1.03
Average lead in matte	13.47	13.0

Recapitulation

The D. & L. and the H. & H. plants are operated conjointly in some respects; indeed, both are supplied with ore from the same cylindrical mixing bins and from the same belt conveying system. One metallurgist and one and the same set of foremen have the responsibility of handling the two types, abundant opportunity is offered for comparison, and absolutely no circumstances exist which would swerve opinion either one way or the other. It is believed that a fair summary of the actual experience set forth in this paper would be as follows:

	Advantage in Favor of
Cost of installation	D. & L.
Cost of roasting	H. & H.
Adaptability of charge	D. & L.
Metal losses	Doubtful
Physical condition of product	H. & H.

Conclusions

The D. & L. is a good modern machine now brought to a state of efficiency where it is splendidly adapted to lead-smelter work. The H. & H. process is still operated with a good measure of success. A good-sized, compact H. & H. plant at Murray, well designed and operated in a painstaking manner, permits of possibly an unusually good expression being recorded in favor of the H. & H.

The low first cost of D. & L. units and the ease and flexibility with which successive units may be installed at any given lead smelter are perhaps the reasons why a greater number of 400-ton H. & H. plants have not received thorough trial.

Actual experience with D. & L. machines at Murray has caused them to gradually grow in favor from year to year. We are highly pleased with the flexibility of charge and other important items. The beautifully sintered product is so exceedingly good as to make it unfortunate that exaggerated claims for its virtues should have been published.

This article is unable to point out any overwhelming advantage of the D. & L. over the H. & H. system, although continued progress may upset the balance at any time. If history repeats itself some new roasting system will take superior rank over both within a few years.

Discussion

Arthur S. Dwight, New York, N. Y.—Mr. Norton's chief argument is directed against the idea that the D. & L. sinter is any better material for the blast furnace than the H. & H. product. He supports this by citing two experimental furnace runs.

Reducibility.—To discuss a furnace record like that quoted by Mr. Norton for the five days' run in August, 1912, is manifestly difficult for an outsider, but a possible explanation for the poor reduction on Furnace No. 5, smelting D. & L. sinter, may lie in the fact that both sets of furnaces carried the same fuel charges. Other things being equal, if 11½ per cent. coke was right for the furnaces smelting H. & H. product, it was too much for the furnace smelting D. & L. product, as I shall endeavor to show later, and incipient overfire, accompanied by higher leads in slag and matte, was induced. It may sound like a paradox to say that excessive fuel sometimes produces poor reduction, but it is a sad fact.

In certain plants where practically the entire charge to the blast furnaces receives a preliminary treatment by being sintered on the D. & L. machines the peculiarities of this problem have had to be very carefully studied, for serious metallurgical difficulties presented themselves when the change was first made to D. & L. sinter in large quantities. Some of these plants had blast pots before they had D. & L. machines, and many opportunities were afforded for comparative study of this very point. There is reason to believe that, due to the long period of blowing and the higher temperatures developed in the pots from greater mass action, the iron in the H. & H. product is present chiefly as Fe_2O_3, perhaps in the form of ferrites, while in the D. & L. product the iron is present as FeO. The fuel charge in the blast furnace must be so proportioned as to exactly meet the chemical and thermic requirements of reduction and melting. If any of these functions have been performed before the material enters the furnace, a corresponding allowance should be made or the fuel balance will be disturbed and bad work will result. One of the functions of the coke is to reduce enough metallic Fe for the matte requirements, leaving the remainder of the iron as FeO to go into the slag. In smelting almost any other material than D. & L. sinter the iron must be reduced from the higher oxides, Fe_2O_3 or Fe_3O_4, to the protoxide, FeO, before a true comparative basis is reached. Experience has shown that with about one-half D. & L. sinter on the charge the coke should be cut about 10 per cent. This theoretical benefit is borne out by the practical blast-furnace results on a large scale in numerous plants. In fact, wherever this very important factor is ignored the evil symptoms of excessive fuel will be indicated, one of which symptoms is a diminished furnace speed with tendency to overfire, and, as every metallurgist knows, this is likely to be accompanied by poor reduction in slags and mattes, especially if the mechanical features of the problem, such as size of lumps and distribution in the furnace, have not been correctly adjusted. This general fact has been quite convincingly proved, at least to my own satisfaction, by the numerous cases where I have been called upon to observe difficulties and recommend remedies in connection with the growing use of D. & L. sinter in smelting operations.

To put the matter tersely, and emphasize the important point I wish to make, the successful smelting of D. & L. sinter not only *permits* of economy of coke, but actually *demands* it.

Furnace Speed.—In the furnace record of Aug. 12 and 13, 1912, when the furnaces smelting D. & L. sinter ran a trifle slower than those with H. & H. product, it is to be noted that the former carried 580 lb. iron ore on the charge, while the latter required 1,140 lb. iron ore, indicating that the D. & L. sinter was itself very high in iron. It is well established that a very basic sinter does not smelt as fast as one in which the proper proportion of SiO_2 and FeO for the slag is approximately maintained. This may or may not be the reason in this case, but it is worth mentioning.

The following figures were furnished me some time ago as giving the comparison of average furnace speeds in a certain plant where the character of the charge sent to the blast furnace was gradually improved by the introduction of modern methods of preliminary treatment. The furnaces in question were 48 by 162 in. (54 sq. ft.) at the tuyères, and the averages were taken over considerable periods of time.

	Average Tons Smelted per Furnace Day	Tons per Square Foot Hearth Area per Day
(1) With Brückner roaster product	180	3.3
(2) With Huntington-Heberlein product	200	3.7
(3) With H. & H. product and ore fines largely removed by screening	227	4.2
(4) With Dwight & Lloyd product and ore fines largely removed by screening	285	5.3

Case (4) shows a gain of 25 per cent. over case (3).

In another Western plant the addition of D. & L. sinter to the charge increased the furnace speed from 140 tons per day to 175 tons per furnace day, a gain of 25 per cent.

In still another case the increase was from 180 tons to 210 tons, an increase of 17 per cent.

These are from the reports of the managers and can be easily verified. Many others could be given.

Low Melting Point.—Mr. Norton expresses his belief "that a partly fused or 'predigested' combination may tend to poor results rather than to good results when smelted, for the reason that such substances fuse at too low a temperature in the furnace."

I fully recognize this as a possibility and indeed a probability when D. & L. sinter is treated without due regard to its properties, but I positively assert that such tendency can be quickly and easily met with no change from ordinary standard furnace equipment, and with distinct fuel economy, as has already been pointed out.

A very pertinent citation on this point is the experience of a certain smelter superintendent who had to carry a considerable amount of leady converter slag on his charge. He was much troubled by premature fusion of this ingredient until he discovered recently that by crushing it and adding it to the charge of his D. & L. sintering machines he was able to control this tendency and is now regularly following this practice.

I do not wish to prolong this discussion unduly and therefore will only suggest in the briefest possible manner my passing comments on the other matters I would like to speak of.

Sulphur Range.—The general experience has been that higher initial sulphurs can be handled by the D. & L. than by the pots. This is to be expected from the smaller mass under treatment at one time, and the mechanical means for rapidly abstracting the heat as it is developed by the oxidation of the sulphur. As an example a certain plant figures its D. & L. charges between 12 and 14 per cent. sulphur, and its H. & H. charges for direct conversion at 10 to 11 per cent. sulphur.

Sulphur Elimination.—There are many data available to indicate that the D. & L. is at least as thorough in this respect as the pots, with a far greater efficiency per unit area of plant.[1]

Operating Costs.—The H. & H. plant at Murray has about double the capacity of the D. & L. plant. The operating cost per ton is necessarily influenced by this fact. If the costs were corrected to an equivalent basis the small difference in favor of the large H. & H. plant would probably be more than wiped out. Without going into details, it may also be said that a true comparison should include all the pertinent items, such as interest on investment, depreciation, etc., which would unquestionably throw the balance in favor of the D. & L. In this connection, I wish to say that the D. & L. process is most distinctly on its mettle at Murray, for Mr. Norton has worked out the H. & H. process to an extraordinary degree of perfection and his practice in that department justly ranks among the very best.

Fineness of Charge.—As Mr. Norton says, the D. & L. machines have to handle the very fine material that the pots cannot treat. In many plants this includes also the fines from the pots. In other words, the machines have to do the dirty work, because they can, but it is only fair to ask how the figures would be affected if the H. & H. pots had to take their share of the very fine material.

Lead Losses.—Mr. Norton does not attempt to present any conclusive data on this point. As the volatilization is influenced largely by the temperature and the time the charge is subjected to the heat and air currents, the advantage ought theoretically to be with the D. & L., for in that process a given particle is undergoing the heat treatment about as many min-

[1] A. S. Dwight: Efficiency in Ore Roasting, *School of Mines Quarterly*, vol. xxxiii, No. 1, pp. 1 to 17 (Nov., 1911).

utes as it is hours in the H. & H. pot. Investigations of lead losses have been made at various plants, some of which will probably be published some day; meantime, it can be stated that while it is possible to establish conditions which will make a D. & L. machine into a very efficient lead burner, it is not necessary so to do, and the lead losses should be less than with the pots. As an example, in one plant under my direct observation where pots have been entirely superseded by D. & L. machines, the lead loss was about 3 per cent. in the pots and is now about 0.75 per cent. as the average over a period of two years with the machines.

In conclusion I wish to again thank Mr. Norton for his interesting paper.

G. C. RIDDELL, East Helena, Mont.—Mr. Norton's interesting paper, comparing the H. & H. and D. & L. processes, no doubt reflects with a high degree of accuracy the behavior of these two roasting methods as he has found it at the Murray plant.

Up in Montana, however, we are working with somewhat different conditions, and as we have been a little more favorably impressed with the relative merit of the Dwight-Lloyd sintered product, it may be of interest at just this time to describe the behavior of these two processes at the East Helena lead plant of the American Smelting & Refining Co.

The reason for a difference in performance of these roasting processes at Helena and at Salt Lake lies perhaps in the rather unique roasting and smelting practice at the former plant, where all operations are on charges carrying an abnormally high lead and zinc content. Helena mixture beds contain approximately 80 per cent. Cœur d'Alêne lead sulphides and average 160 to 180 lb. to the cubic foot. The local significance of this high percentage of heavy leady material can readily be seen when it is remembered that the processes under discussion involve the blast-roasting principle. This density, together with the ready fusibility of such a lead-bearing charge, presents a set of conditions at East Helena that give rise to a number of interesting problems at both roasters and blast furnaces.

Certain it is that with both H. & H. and Dwight-Lloyd products in excellent physical condition, our East Helena blast furnaces run at higher speeds and with distinctly lower coke when dominated by the Dwight-Lloyd sinter. All those daily observations, recorded and unrecorded, that the metallurgist makes as he guides his blast furnaces through their various ailments and difficulties, have for several years past confirmed us at East Helena in the feeling that the more Dwight-Lloyd material we can get on our furnaces in place of H. & H. product, the lower is the percentage of coke required and the faster do the blast furnaces run. That we have dealt with extremes as well as averages in these matters, may be seen from the fact that the East Helena furnaces have carried upward of 90 per cent. roast on charge for long periods, with smelting tonnages of 475

to 575 tons per 100 sq. ft. tuyère area. An average of 249 tons per furnace day, maintained during the month of March, 1913, figures 549 tons per 100 sq. ft. tuyère area.

In December, 1912, one blast furnace was run for seven days on a charge containing H. & H. roasted product and no Dwight-Lloyd; a furnace alongside of it running on Dwight-Lloyd product with no H. & H.—the idea being to note the tendencies of the two roasted products as to speed and reduction. The composition of the two charges used is shown in the following table:

	H. & H. Charge, Pounds	Dwight Machine Charge, Pounds
Siliceous bed ore	1,580	176
H. & H. roast	5,264	....
Dwight sinter	.....	7,040
Lime rock	1,076	322
Scrap iron	80	80
Golden Curry iron ore	.....	382
	8,000	8,000

The results of the week's test are shown in the tabulation on the following page and indicate that 2 per cent. less coke on charge, better reduction, and higher speed attended the use of Dwight machine product. It is to be noted that the Dwight machine charge was run part of the time on one furnace and part of the time on the other, in order to make a fair comparison of the Dwight-Lloyd and H. & H. products regardless of local furnace conditions. In every instance except one, the Dwight-Lloyd product gave a more favorable showing than the H. & H. This exception is indicated in the wet lead in the slag on furnace No. 2, the result here being distorted by hang-over assays, after the H. & H. charge had been changed to furnace No. 1. Eliminating these hang-over assays, the slag wet lead on furnace No. 2 for the H. & H. charge becomes 1.37 per cent. and for the Dwight 1.29 per cent.

It was found, in fact, that with furnace No. 2 doing poor work on the H. & H. charge it was possible to straighten out both the tonnage and assays by switching to the Dwight charge. In all cases we had good crucibles, bright tuyères, and cool tops, but it was found necessary repeatedly to take off considerable amounts of coke on the Dwight furnace, on account of persistent over-reduction.

It was found in general that our mechanical feeding arrangements were more favorably adapted to the proper distribution of the Dwight machine product. As fast as they were indicated, however, changes were made

Huntington and Heberlein

	Dec. 5 to 9, 1912	Dec. 3, 4, 1912	Average
Furnace, No.	1	2	
Per cent. coke	13.75	12.50	13.39
Per cent. F. C.	10.9	9.8	10.6
Tons wet charge	210	208	209
Slag:			
Per cent. zinc	5.0	4.9	5.0
Ounces silver	0.29	0.26	0.28
Wet lead	1.99	1.47	1.84
Per cent. lead on charge	30.1	30.4	30.2
Per cent. roast	66.2	67.9	66.7
Per cent. sulphur on charge	4.2	4.2	4.2
Per cent. matte fall	18.3	16.8	17.9
Matte:			
Lead	24.6	18.8	22.9
Copper	9.9	10.9	10.2

Dwight and Lloyd

	Dec. 3, 4, 1912	Dec. 5 to 9, 1912	Average
Furnace, No.	1	2	
Per cent. coke	11.57	11.39	11.52
Per cent. F. C.	9.0	9.0	9.0
Tons wet charge	216	224	218
Slag:			
Per cent. zinc	5.1	5.1	5.1
Ounces silver	0.21	0.25	0.22
Wet lead	0.72	1.48	0.94
Per cent. lead on charge	30.6	30.2	30.5
Per cent. roast	88.8	87.6	88.4
Per cent. sulphur on charge	3.7	3.8	3.7
Per cent. matte fall	15.4	15.9	15.6
Matte:			
Lead	9.8	17.8	12.1
Copper	8.1	9.4	8.5

in the distribution of both charges, and it was felt that each one of the changes made was in the nature of an improvement at the time. The test was too short, however, to permit these changes in feeding arrangement to proceed to a point at which the H. & H. product was distributed to absolutely the best advantage. In fact, had we been able to prolong the test, we could undoubtedly have made a more favorable showing for the

H. & H. by further changes in feed-floor arrangement. It is true, however, that the over-reduction on the Dwight furnace prevented the large tonnages that were clearly indicated for the Dwight charge under proper conditions.

It should be noted that in these tests both charges involved larger percentages of roasted product than were used in the Murray trials—65.8 and 88 per cent. against 37.5 and 60 per cent.

In Mr. Norton's recapitulation, five items are set down, two in favor of H. & H., two in favor of Dwight-Lloyd, and one with honors even. These same five items viewed in the light of East Helena operations compare as follows:

1. *Cost of Installation.*—Regardless of location, this is, of course, in favor of the Dwight-Lloyd apparatus.

2. *Cost of Roasting.*—During the year 1913, the Dwight-Lloyd plant at East Helena accomplished a sulphur elimination of from 10.9 to 4.6 per cent., while the H. & H. converter pots during the same period reduced from 10 to 5.1 per cent. sulphur. In this narrow range of su'phurs is seen the deterrent effect of the heavy dense lead charge at East Helena on both methods of blast roasting. The roasted material at present turned out in the H. & H. department averages 38 per cent. lead, while the Dwight-Lloyd sinter carries 43 per cent. lead. The H. & H. process is also favored to the extent of receiving the coarser Cœur d'Alêne products, while the Dwight-Lloyd beds take care of the slimes and fine concentrates. The average figures for the 12 months of 1913 show a cost per unit of sulphur eliminated, the same within a fraction of a cent, for both processes. Three hundred tons per day are handled in the H. & H. plant and about 400 at the Dwight machines.

3. *Adaptability of Charge.*—As at Murray, we recognize a greater leeway on the Dwight-Lloyd process in the physical and chemical variations possible in the charge.

4. *Metal Losses.*—Repeated tests on East Helena flues indicate, as at Murray, that lead losses are practically the same on both processes.

5. *Physical Condition of Product.*—An abundance of experience and testimony concerning the important part in reduction played by ascending blast-furnace gases makes it seem entirely logical and proper to us at East Helena to attribute the beneficial effect of Dwight-Lloyd product on our smelting procedure to its peculiar cellular structure, allowing the reducing action of the gases to proceed over a maximum area of contact from the moment the charge begins its downward journey from the furnace top.

The danger from the fusing of such predigested material (either Dwight-Lloyd or H. & H.) at too low a temperature in the furnace, should be and is offset, at East Helena at least, by the faster descent of the charges containing these roasted products. Although 85 to 90 per cent.

of our smelting stock is made up of roasting product (50 per cent. Dwight-Lloyd, 35 per cent. H. & H.), hot tops are practically unknown.

Recent years have found us gradually and steadily increasing the amount, and improving the character, of the roasted product on charge, at East Helena, and with each step in the process better blast-furnace results have been noted. In short, it is to the roasters and not the blast furnaces that we are inclined to give credit for accomplishing a notable improvement in lead-smelting speed and general metallurgy during the last few years all over the country, and the Dwight-Lloyd process has played such an important part in this at East Helena that we are glad to add our word to the approval expressed by Mr. Norton. In going a little further in our appreciation and acknowledging a distinct superiority of Dwight-Lloyd over H. & H. product, we fully realize that the handling of 35 to 40 per cent. lead charges may present a rather extreme smelting procedure, and that these opinions as to the relative merits of Dwight-Lloyd and H. & H. roasted products may simply reflect local Montana conditions.

A summary of Murray and East Helena experience with these two roasting processes would appear, then, as follows:

	Advantage in Favor of	
	(at Murray)	(at East Helena)
Cost of installation	D.-L.	D.-L.
Cost of roasting	H. & H.	Even
Adaptability of charge	D.-L.	D.-L.
Metal losses	Even	Even
Physical condition of product	H. & H.	D.-L.

Nodulizing Blast-Furnace Flue Dust

BY LAWRENCE ADDICKS, DOUGLAS, ARIZ.

(Salt Lake Meeting, August, 1914)

SOME three years ago the smelter connected with the Chrome, N. J., refinery of the United States Metals Refining Co. found itself embarrassed by constantly increasing piles of unsmelted blast-furnace flue dust. The charge was fine and 10 to 15 per cent. of it was blown over into the flue. Sulphur was at that time too scarce to make the sintering of the finer part of the charge attractive, and raw-ore smelting in a reverberatory with a partly oxidized charge was not to be thought of.

A great many schemes were considered, including leaching, blowing into the converters, and briquetting, the problem finally narrowing down to blast roasting and nodulizing. It was found by experiment that blast roasting would not yield a satisfactory sinter without the addition of coke dust or sulphide fines to augment the fuel value of the flue dust, and on this account, together with the low first cost of a kiln, the latter was decided upon.

A couple of tons of flue dust were tried out in the experimental kiln in the laboratory at Yorktown, Va., of J. H. Payne, who acted as consulting engineer in this connection. This kiln was approximately 2 ft. in diameter and 20 ft. long and was fired with fuel oil. A run of several hours yielded excellent nodules. The kiln showed no tendency to build in and form "nose rings" and gave a fuel consumption of 50 gal. of oil per ton of flue dust. There was no indication of any stack loss. The roasting was strongly oxidizing. A study of what full-size kilns were doing on cement clinker and on nodulizing pyrites cinder indicated that a 60 by 6 ft. kiln would certainly not consume over half of the oil per ton of flue dust shown on the test and as fuel oil at that time was selling around 2 1/2 c. per gallon it was decided to let it go at that.

A 60 by 6 ft. kiln was decided on, as that size is standard in cement practice, although fast being replaced by much larger ones with their greater fuel economy, and as it fitted in the building space available. A 6-in. brick lining brought the net diameter down to 5 ft. The inclination

toward the discharge end was fixed at 5/8 in. to the foot and the revolutions per minute at 1 1/2.

The results, while satisfactory, were quite different from those anticipated. In the first place, the fuel consumption was far lower than had been expected. A granular sand can be made with perhaps 8 gal. per ton, a first-class smelting product with 12, and great chunks with 16. It appears, therefore, that such a test kiln as that used takes about four times the fuel that will be required on a 60-ft. installation, although this ratio might be changed for different materials with varying internal fuel values.

In the matter of formation of nose rings, the test kiln was deceptive. There was a decided tendency to such formations and it took some time for the operators to acquire the necessary skill to control this. Steady conditions of flame are very necessary. If the kiln is overheated semi-molten material forms on the walls and a subsequent over-chilling will plaster the sand on very rapidly. A number of devices were tried to meet this difficulty, but finally it was found that reasonable skill and care on the part of the attendant and an occasional shutdown of a few hours to remove any obstinate obstruction were the best remedies. A patented scheme using a hollow drill, delivering compressed air or water, gave indifferent results, although it seems to have given satisfaction in pyrites-cinder work. Next the accretions were allowed to grow in such a way that they collapsed by their own weight. This, however, took about as much skill as avoiding their formation and, further, greatly diminished the capacity of the kiln. When perfectly clean 75 tons of flue dust can readily be nodulized in 24 hr., while a choked-up barrel will deliver less than half of this quantity. There is no difficulty in regular work in delivering 50 tons a day, including all delays, and a few hours' work for two or three men once in two weeks will handle the accretions. Another experiment that was tried was the use of a boring bar every hour or so while the kiln was in operation, but here again, while the scheme worked fairly well, the cure was worse than the disease. The flame had to be temporarily cut off and the loss of heat resulted in bad product for some minutes after each attack. Finally it was found that spraying with a hose would cause disintegration of these lumps. The nose rings, however, proved to be more of an annoyance than a serious difficulty and caused but little increase in the operating costs.

The normal operating condition of the kiln is to have the 40 ft. next the flue acting simply as a preheater, without any real nodulizing action. The material then begins to stick and ride around, which is the first step toward the formation of nodules. The actual nodulizing should be completed in the next 15 ft., leaving the last 5 ft. for hardening the nodules in the cooler zone back of the flame. This is to keep the nodules from sticking together in a semi-fused mass when they drop into the receiv-

ing car. The nose rings will be confined to the 15-ft. nodulizing zone, where they are readily accessible from the front end when the fire hood is rolled out.

It was expected that delivering red-hot nodules to the blast-furnace would result in quite a saving, but it was found that they smelted so fast that they tended to run away from the rest of the charge and cause uneven operation; much better results were obtained by feeding them cold. This would probably not be true in the case of nodulized ores, where the product would not be self-fluxing.

No trouble was experienced from flue dust blowing out of the kiln. In dry process cement work, where 95 per cent. of the material fed will pass a 100-mesh screen, but 3 per cent. flue dust is made. In the case of our flue dust fed, about 25 per cent. would pass 100 mesh.

Shortly after the kiln was installed the fuel-oil market ran away, the top prices reaching nearly 6c. a gallon; consequently some experiments with coal firing were tried. A large fire box was built, along the lines of those used in reverberatory work, operating largely as a gas producer, and equipped with forced draft. This was an absolute failure, due to the impossibility of keeping an even temperature at the mouth of the kiln. Every time fresh coal was fired the nodules would dwindle down to sand. It is obvious that the same objection would apply to any gas-producer plant not of sufficient size to give a uniform quality of gas. The great majority of rotary kilns are fired with pulverized coal. This requires a fire-box temperature sufficiently high to get satisfactory ignition, and as this flue-dust nodulizing is distinctly low-temperature work it was questionable whether the considerable investment in coal drying and pulverizing would have been justified. Fuel oil is the ideal fuel for such work wherever its cost will allow it to be considered at all.

While the oxidizing action of the small test kiln was very marked, the sulphur being almost totally removed, it was much less so in the larger kiln. Rough grab samples of the entering flue dust and the nodules produced showed the following analysis, which may be taken as representative:

	Flue Dust	Nodules
Au, ounces per ton	0.12	0.10
Ag, ounces per ton	11.30	10.90
Cu, per cent	12.00	13.02
FeO, per cent	31.33	36.93
Insoluble, per cent	22.10	26.50
SiO_2, per cent	18.19	20.80
CaO, per cent	0.48	0.54
Zn, per cent	1.97	1.88
S, per cent	11.86	4.95

It will be noted that but little over half of the sulphur has been removed. There is also some loss in silver indicated, but the sampling is too rough

to permit drawing any conclusion. No special attention was paid to silver losses, as the material treated was all low grade, but rich fines would doubtless show a heavy loss, judging from the results obtained on the old Brückner roaster in the early days.

Physically the nodules are an ideal blast-furnace product. A screening test when making quite small nodules gave the following results:

Between	Per cent.
1/2 and 1/4 in.	54.21
1/4 and 3/16 in.	23.16
3/16 and 1/16 in.	20.60
1/16 and 20 mesh	1.10
20 and 30 mesh	0.54
30 and 40 mesh	0.14
40 and 60 mesh	0.12
60 and 80 mesh	0.05
Through 80 mesh	0.08
	100.00

While nodulizing is comparable with blast roasting, the two processes have different limitations, much as in the case of a comparison between blast furnace and reverberatory smelting. Sintering requires a properly balanced internal fuel supply, while nodulizing is entirely independent of this. To properly nodulize, a material must have a sufficient interval between the temperature of becoming sticky (incipient fusion) and that of actual melting to give a practical range of operating flame temperatures. As a nodule is formed from individual grains stuck together, every piece has a solid structure and there are no defective ones full of unsmelted material. I believe the kiln is deserving of a somewhat wider application in the metallurgical field than it has as yet obtained, but the extent of this expansion can only be determined by experiment. While these notes cover but a single material, blast-furnace flue dust, and that from but a single plant, yet this material is much more like ore than pyrites cinder and other refractory products. It would be advisable to use a kiln 7 or 8 ft. in diameter to lessen the nose-ring difficulty, and this is in line with the experience in other classes of work. As to length of kiln for low-temperature work, 60 ft. is probably long enough; for refractory materials calling for a greater unit fuel consumption longer kilns will pay in fuel saving.

As far as operating cost goes, the crux of the whole matter lies in the cost of fuel. A single operator can attend to the firing of several kilns; and the repairs should be very small, as the machinery is very simple, thoroughly standardized, and not exposed to the heat.

Discussion

James H. Payne, Baltimore, Md. (communication to the Secretary*). —Mr. Addicks's account of the behavior of the nodulizer at Chrome on blast-furnace flue dust leads me to outline what has been done so far upon the application of the same apparatus to fine sulphide concentrates.

As Mr. Addicks states in his paper, the conditions prevailing in the test kiln at Yorktown in the demonstration I made for him upon Chrome material were strongly oxidizing; so much so that the sulphur was roasted down to less than 1 per cent. It is therefore to be expected that the first test made upon sulphide ore a short time after this, in which Ducktown green ore fines were used, did not show very promising results. Although the Ducktown ore contained only 13.52 per cent. of sulphur its gangue was quite fusible, and the intense heat generated by the oxidation of the extremely fine sulphide particles slagged the mass, bringing about a condition approximating that of a Brückner roaster in distress.

During the spring and summer of 1912, the writer kept working upon different schemes to cut down this slagging. Two successful methods were finally worked out embodying opposite principles, one promoting oxidation of the finer particles and the other retarding their oxidation.

In the early part of the present year the writer succeeded in interesting the Braden Copper Co. in nodulizing, with the result that an extensive series of tests were made at Yorktown upon oil-floated concentrates covering quite a wide range in analysis. These tests led to trials in the Chrome rotary, on the part of the U. S. Metals Refining Co., of Minerals Separation concentrates which they had in stock, and later to test runs to check up the Yorktown runs on Braden concentrates. The large-scale tests checked up with the Yorktown tests in every way (except fuel consumption, which was, however, correctly predicted), and led to the Braden company's decision to adopt the process at Braden.

The fuel consumption upon oil-floated concentrates is not over 6 gal. per ton and in many cases is less. The action in the furnace is independent of the sulphur content, and there is actually less tendency to form nose rings in the Chrome rotary than upon flue dust. It is believed that the improvements entering into the design of the Braden kilns will cut down the nose-ring trouble to where it will no longer be a drawback to the process.

The flue dust produced by the kiln itself is practically *nil*, although some of the material treated has been as fine as 82 per cent. through 100 mesh. This is unbelievable to metallurgists familiar with the old Brückner roaster; but the conditions are entirely different. The material is fed wet and what dust there is precipitates in the atmosphere of steam at the exit end. The material, furthermore, is constantly moving down-

* Received Aug. 5, 1914.

ward to the hot end and is in a dry, dusty state but a short time, soon changing to a densified condition which is no longer dust.

The amount of sulphur in the nodules is under perfect control. It can be as high as 15 per cent. or as low as 5 per cent., as may be desired. The upper limit for good working seems to be about 15 per cent., as the nodules are very sticky with higher sulphur content. A product running 13 to 15 per cent. sulphur has been readily obtained on all oil-floated concentrates so far tried, where the object was to retain all sulphur possible. This makes pyritic blast-furnace smelting of sulphide concentrates possible.

The size and character of the product has varied with the different concentrates tested so far, and also with the sulphur content sought for. When high sulphur is desired in the product the nodules run smaller than the flue-dust nodules as produced at Chrome. They are, however, entirely free from material that would blow out of a blast furnace. A number of samples tested show from 42 to 45 per cent. of voids.

Nodulizing of sulphide concentrates, particularly oil-floated concentrates, opens up a new field, because it makes pyritic smelting of such material possible. Further, the operation appears to be such a cheap one that it may compete with roasting in multiple-hearth furnaces in reverberatory practice. If so, it would be far preferable to roasting in the case of the exceedingly fine oil-floated concentrates, because of the large amount of dust that the roasters must necessarily produce on this material.

R. M. DRAPER, Chrome, N. J.—The kiln will treat El Cobre flotation concentrates very successfully. There does not seem to be much of a tendency to form ring accretions, and those that form are nearer the discharge end of the kiln, where they may be removed much more readily. The temperature of the kiln is much lower for concentrates than for flue dust and the oil consumption much less. Our consumption of oil was about 6 to 7 gal. per ton of concentrates.

The nodules were very satisfactory from a blast-furnace point of view when charged into the furnace cold. We did have considerable trouble with crusts on the furnace if we put in too much of the hot nodules, due probably to the fact that they were near the smelting point when charged into the furnace and naturally smelted much higher up in the furnace. I think we would have had this difficulty with any hot material that was charged. In fact, we found the same tendency to crust when we tried the experiment of charging hot converter slag into the furnace several years ago.

The concentrate feed averaged about 30 per cent. sulphur, and several samples of nodules taken at various times showed an average of 14 per cent. sulphur. One sample of nodules chilled immediately in water showed a sulphur assay of 20 per cent.

There is no difficulty in averaging 60 tons of concentrates per day, and I believe that with a proper feeding device this tonnage could be increased to 75 tons per day.

There is no question that the greater the tonnage the less consumption of oil per ton.

Screen Test on Nodules from El Cobre Concentrates

	Per Cent.
Oversize (3 in.) and ¼ mesh	25.95
Between ¼ and 8 mesh	44.75
8 and 16 mesh	24.52
16 and 20 mesh	1.64
20 and 30 mesh	1.93
30 and 40 mesh	0.46
40 and 60 mesh	0.33
60 and 80 mesh	0.14
80 and 100 mesh	0.04
100 and 120 mesh	0.06
Over 120 mesh	0.18

Smelting Lead Ores in the Blast Furnace

BY IRVING A. PALMER, EASTON, PA.

(Salt Lake Meeting, August, 1914)

DURING the past 15 years in this country there have been few additions to the literature of lead smelting. After the consolidation of the principal smelting companies at the beginning of this period it became the policy of the larger company not to permit the publication of details in regard to many of its metallurgical processes. This led to similar action on the part of some of its competitors. The result, of course, was a decided decrease in the number of papers written upon the practical side of the subject. Even upon the theoretical side the contributions have been few. It would seem that the industry as a whole has suffered from this general unwillingness to exchange information. While it is true that a given company is entitled to the reward that may come from its own development of a new process, it is also true that a still greater reward might be the result of a free exchange of ideas with other plants. This principle has been recognized for a long time in other metallurgical industries, notably those engaged in the smelting of iron and the manufacture of steel. It is pleasing to note that the leading lead-smelting company has very recently decided to modify its policy of secrecy, and to permit the publication of papers covering some details of its metallurgical methods.

The statement has been made that there has been but little progress in lead smelting during the period just referred to. This impression is due in part, no doubt, to the scarcity of papers written on the subject. There has been, it is true, some discussion of the new sintering processes, but with little reference to detail, and to the effect that blast roasting has had upon the industry as a whole. In spite of the prevailing belief, it may be said that lead smelting, in common with other metallurgical industries, has made steady and substantial progress.

It is the purpose of the present paper to discuss briefly some features of the industry as it is now conducted, with special reference to the blast-furnace practice in the larger silver-lead smelting plants of the United States and Mexico.

It is generally conceded that the greatest advance made in lead smelting in recent years has been effected by the introduction of the various sinter-roasting processes. The past decade has witnessed the practical abandonment of hand roasting, and of mechanical roasting in which the product is not eventually sintered. As will be shown later, this improvement has greatly increased the efficiency of the blast furnace, both as regards tonnage and as to the character of the work done.

Second in importance has been the general adoption of mechanical methods in the handling of materials. The labor problem has always been a serious one in the industry. Owing to the comparatively small tonnages, the great variety of ores, the necessity of careful bedding and mixing, and the delicacy of the various metallurgical processes, it has never been possible to do away with hand labor to the extent that it has been accomplished in iron and copper smelting. But a great deal has been done in this direction, and further improvements are being considered.

The agitation in many localities against smelter smoke, as well as a more careful study of fume losses, has led to the construction of bag houses at a number of smelting and refining plants. These bag houses have not only effected a saving of values, but have thrown considerable light upon the whole subject of metal losses. They have added new problems as well, because of the large amount of volatile by-products collected by them.

There has been, also, an increase in the size of the blast furnaces and of other equipment employed in lead smelting. The increase has been moderate, however, and there have been no radical changes, either in size or in general arrangement. In this connection it may be said that the opportunities for experimenting along these lines by the average metallurgist are limited. Most companies are unwilling to stand the expense of conducting experiments on a large scale. Even when the money is forthcoming the difficulties in the way of testing properly a large blast furnace, for example, are great. It has been the general practice to increase the size of blast furnaces at new plants, or at plants that are being rebuilt, and in these cases the other equipment can be constructed to conform.

Let us now consider some of the details of present-day practice:

The blast furnace is the central feature of a smelting plant. The buildings and equipment should be constructed with the idea of serving the blast furnaces economically and efficiently. That this is not the case in some plants is a well-known fact, and is partly responsible for the reputation for conservatism which the industry has.

As intimated above, there has not been much change in the construction of the lead blast furnace in recent years. At the present time the maximum length of the furnaces that are used in this country is 200 in. at the tuyère level, and the maximum width, 48 in. At most of the large

smelting plants the length varies from 136 to 160 in., and the width, from 42 to 46 in. So extreme a width as 62 in., as in some foreign countries, has not found favor in the United States. A width of less than 48 in. is preferred by some because of their belief that hotter slags are thus produced.

The angle of the bosh has been the subject of considerable controversy, as well as of experimenting. With the furnace charge constituted as at present at most plants, the weight of experience is decidedly in favor of a moderate bosh. At a plant with which the writer is familiar the furnaces were built with a 20° bosh, so that the width at the top of the 6-ft. jackets was 84 in., or 36 in. greater than at the tuyère level. Whenever a furnace was blown in, and for some time thereafter, it was noticed that there was a tendency for the charge to hang at the bosh, and to descend irregularly. The assays of the slags and mattes were usually higher than those of the furnaces longer in blast. After a week or two the furnace work would begin to improve, and later would become normal. On blowing out a furnace, after a campaign, it was almost invariably observed that the space on each side of the furnace above the bosh had been filled in with partly-fused charge, changing the furnace lines so as to form a gently flaring shaft 60 in. wide at the top of the jacket, or with a flare of about 1 in. to the foot on each side. The furnace had thus accommodated itself to the descending charge, and, as stated, better work was done as a consequence. The extra width of the furnace above the bosh was not only useless, but detrimental. With the furnace charges as they are now prepared there is but little contraction in volume as they descend in the shaft until the fusion zone is reached. There is also a much more rapid movement of the charge. In addition, the narrower shaft insures a better distribution of the ingredients of the charge, and as a rule produces hotter slags.

The tuyères are usually spaced 16 in. apart from center to center and have openings varying from $2\frac{1}{2}$ to 4 in. in diameter. On this point of tuyère area there is considerable difference of opinion. With coarse charges, wide furnaces, and the comparatively low blast pressures now used, there is a tendency to employ a smaller tuyère opening, in order to give a better penetration to the blast. With a tight charge and high blast pressure the question of tuyère area is not so important.

The proper height of the charge column is a matter that depends very largely upon the physical character of the materials of which it is composed. There has been a reaction from the low column formerly used, because of the flue-dust and fume losses, and of the excessive speed, which often results in foul slags and mattes. On the other hand, a very high column has been found to decrease the furnace speed and to increase the tendency for the fire to creep up in the shaft. In the large furnaces now in use the charge is usually maintained at a height of from 15 to 18 ft.

The water jackets are now commonly made of soft steel. While

their cost is greater than that of cast-iron jackets, they last longer and can be easily repaired when damaged. To avoid burning, caused by the accumulation of mud and scale in the bottom of the jackets, they are washed out at regular intervals. In a late improvement the bottom of the jacket, instead of being horizontal, is inclined at an angle toward the outside of the furnace, thus throwing the scale and mud away from the fire sheet. Burned jackets are often the result of irregularities in the furnace work.

At the larger plants hand feeding of the furnaces has been generally superseded by mechanical feeding; that is, the charge is dropped from a car which is run over the furnace top. The car discharges either from the center or from each side. In either case the furnace is provided with what are known as "spreaders." These spreaders serve to throw the charge toward the sides of the furnace, thus distributing a large proportion of the fine ore along the sides, while the coarser parts of the charge roll toward the middle line of the furnace. The natural tendency of the blast being to follow the sides of the shaft, it is counteracted in this way. There are few things in lead smelting that have given the metallurgist more concern than this point of securing a proper distribution of the charge in the furnace. It is difficult to secure the right adjustment. Often it can be obtained only after repeated trials. To get the best results there must be taken into consideration the character of the charge, the size and shape of the furnace, the height of the charge column, and the construction of the car. It is one of the proofs of the sensitive nature of lead smelting that a slight modification of the spreading devices often affects profoundly the tonnage and efficiency of the furnace. Improper feeding often neutralizes the effect of good work in all the other departments of the plant.

The preparation of the furnace charge is an important feature in the operation of a lead-smelting plant. Every lead metallurgist knows that the efficiency of the blast furnace is very largely dependent upon the physical character of the charge. It is a difficult problem to prepare a charge that will smelt rapidly and also produce clean slags and mattes, and that is why it is so difficult to introduce satisfactory methods for the mechanical handling of ores and by-products. It is this that makes lead smelting so widely different from iron and copper smelting.

The elimination of the greater portion of the fines from the furnace charge has long been considered one of the essentials of good work. In order to avoid the evil effects of a tight charge it was formerly the custom at some plants to screen the oxidized ores, and then briquette the fines with flue dust, slimes, concentrates, and fine roasted ore from the mechanical roasters. The product of the hand roasters ordinarily was not briquetted. But briquetting is an expensive process, and the briquettes made were often so friable that they fell to powder in the upper part of the blast

furnace. However, something had to be done with the excessive amount of fine material that had to be smelted, and briquetting was considered a necessary evil. The practice undoubtedly increased the furnace speed and efficiency, but it was often carried to excess.

As noted at the beginning of this paper, probably the greatest advance made in lead smelting in recent years has been the general introduction of blast-roasting processes, such as the Huntington-Heberlein and the Dwight-Lloyd. In these processes roasting and sintering are accomplished in one operation, giving a product that greatly increases the capacity and efficiency of the blast furnace in which it is smelted. Sintering pots and machines have largely replaced the old-fashioned roasters and the briquetting presses. The common practice is to sinter a mixture consisting of oxidized fines, siliceous sulphides, concentrates, slimes, and pre-roasted high-sulphur ores. Almost any fine ore can be used, provided the sulphur of the entire mixture does not exceed or fall below a certain figure, which usually varies from 12 to 14 per cent. Ordinarily only high-sulphur ores and lead mattes are subjected to a pre-roasting. In the latest practice this is done in Wedge mechanical roasters, in which a large tonnage can be roasted to almost any desired degree of oxidation, using practically no fuel. The sintering processes enable the lead metallurgist to handle cheaply and satisfactorily nearly all the classes of fine material with which he has to deal, except flue dust. This latter product usually interferes with the sintering process, and, in the opinion of the writer, should be briquetted. If a smelting plant be properly conducted the amount of flue dust made will be small. By using a mixture of roaster and blast-furnace flue dust usually a self-binding material can be obtained. It is also sometimes advisable to briquette high-grade concentrates, rather than to subject them to the sintering process. It is hardly necessary to condemn the practice of simply wetting flue dust and then charging it into the blast furnace. Yet this was done only a few years ago at a large plant in Mexico.

The advantages of the sintered ore as a material for the blast furnaces are to be found in its coarseness, its porosity, and in its similarity in composition to the general average of the charge. The latter is an important point. It is often held that the detrimental effects of fines on the charge are largely mechanical, such as decreasing the furnace speed, increasing the amount of flue dust made, and causing the formation of accretions and blowholes. As a matter of fact, one of the most serious objections to fines is that they cause irregularities in the formation of the slag and matte. The fines of a charge naturally precede the coarse material in getting to the lower part of the furnace. This is due both to the natural fall through the openings in the charge and to the jigging effect of the blast. This jigging action is sometimes considerable. When the first furnace of one of the Mexican smelting

plants was blown in a few years ago the blast was supplied by a blower displacing 300 cu. ft. of air per revolution. Owing to the size of the blower it was run much below its normal speed, causing the blast to be delivered in slow pulsations. The furnace charge descended so irregularly that a satisfactory slag could not be made, and the metal losses were high. When more furnaces were blown in and the speed of the blower increased the furnace work rapidly improved.

The metallurgist computes the slag and matte composition from an analysis of the entire charge. In order that the slag and matte may be formed quickly and uniformly it is necessary that all of the constituents of the charge should be well mixed and arrive at the fusion zone of the furnace at approximately the same time. If they do not arrive at the same time it is evident that at any given moment the amount of some ingredient of the slag and matte to be produced will be either too great or too small. As nearly as possible all of the reactions involved in the slag and matte formation should take place simultaneously. The heat due to these reactions is thus evolved in the shortest possible time. Irregularities are almost sure to occur when the fines of the charge differ greatly in composition from the rest of the mixture, as in the case of lead carbonates and sulphides, raw iron sulphides, and limonite ores used as flux. Finely divided lead ore is particularly bad; much of it is imperfectly reduced and thus goes in part into the matte and slag. On the other hand, a portion of the lead carbonate fines is reduced too soon, and the upward rush of the blast carries small particles of metal to the upper part of the furnace, when they again combine with oxygen or sulphur, forming accretions, or going into the dust chamber. Shaft accretions always contain a large proportion of lead sulphide, even though the charge be almost free from galena. If the fines consist largely of lead and zinc sulphides the reduction is unsatisfactory. Zinc sulphide is considered to be about the worst mineral with which the lead metallurgist has to deal. It is the only sulphide, ordinarily met with, that does not alloy in all proportions with the other sulphides found in lead mattes. When present in considerable quantity it floats on top of the matte, forming a pasty scum and preventing a good separation of the matte and slag. Poor roasting is almost invariably followed by poor work in the blast furnace. Insufficiently roasted ore usually contains an undue proportion of lead and zinc sulphides, owing to the fact that, in any roasting process, these two sulphides are the last to be oxidized. The writer is familiar with a case where a pyrite concentrate containing a large percentage of zinc was subjected to a preliminary roasting in a Wedge furnace sufficient to oxidize nearly all of the iron. An analysis of the product showed that over 90 per cent. of the zinc contents remained as sulphide. Tests have shown that even in well-roasted Dwight-Lloyd sinter, low in zinc, nearly half of the zinc is present as sulphide. As the fines produced in the sintering processes contain most

of the unaltered sulphides, it can be seen how important it is to keep them as low as possible.

The reduction of lead and silver from sulphides is effected mainly by metallic iron and by the well-known reactions between oxides and sulphides, and between sulphides and sulphates. The direct reducing action of the coke is small, except in the case of carbonates. It is evident, therefore, that to obtain the best reducing action the lead and silver bearing portion of the charge must be in close proximity to all of the other principal ingredients. At one of the Mexican smelting plants some years ago the furnace charges were made up largely of coarse siliceous ores, coarse lime rock and limy ores, and iron-lead carbonates containing a large percentage of fines. Blast-furnace flue dust, raw lead concentrates, and pyrite ores were frequently added. The smelter had no sintering plant and no briquetting press. The charge as a whole was very open, as was indicated by the low blast pressure for a given volume of air. Yet the tonnage smelted was low and the lead losses high. The furnaces ran irregularly and there were continual crucible troubles. The fines of the lead and iron ore, the concentrates and the flue dust, sifting down through the charge, were the first to reach the fusion zone, thus destroying the adjustment of conditions necessary to good work. It was found that by crushing the lime ores and adding siliceous fines to the charge a material improvement in the results could be effected. In Pueblo, Colo., at one time it was a common occurrence for finely divided Cripple Creek ore to pass unfused into the slag. The fines were refractory and fed in large quantities. The furnace tonnage was low. It should be stated that careful assays showed that the gold tellurides had been thoroughly leached out of the unsmelted ore.

The best illustration of the disadvantage of using fines is to be found in the present practice in matte concentration. Formerly it was the custom at all plants to calcine the first matte in hand roasters. The product was rather fine, although pretty free from dust. The charge for concentrating this to a high-grade matte was prepared by mixing the roasted matte with coarse siliceous ore and coarse lime rock. Chemically considered, the charge could not have been better, yet the furnace work was poor. The tonnage was usually low, and the mattes and slags high in lead. There was a considerable reduction of metallic copper, with resulting crucible troubles. Often no attempt was made to keep the lead-well open, and the lead was run out with the slag and matte. Sometimes the anomaly would be presented of a considerable formation of speiss, in spite of poor reduction, as indicated by the assays of the slag and matte. The fine roasted matte descended irregularly through the charge and was partly reduced before it could react properly with the silica and lime rock. The present practice in matte concentration at the more advanced smelters is to preroast the matte and then sinter it with

siliceous fines. This gives a coarse, fairly neutral product which, with the lime rock and siliceous ore, descends uniformly in the furnace and causes no segregation at the fusion zone. The result is that the speed and reduction of the matte-concentration furnaces are often better than is the case with the furnaces smelting ore. Using sintered matte the writer has seen concentration furnaces run steadily month after month, producing clean mattes and slags, making but little speiss and metallic copper, and showing no crucible troubles.

There must be a certain amount of fine material on the charge, as otherwise the charge will be so open as to cause too rapid smelting and insufficient reduction. The fines should be of the same general composition as the rest of the charge. Segregation of coarse and fine material would not then result in irregularity. Sufficient fines for this purpose are usually obtained in connection with the various sintering processes.

The larger pieces of the different ingredients of the charge should be crushed to a fairly uniform size. Oxidized ores are now ordinarily sampled mechanically, which necessitates previous crushing. Lime rock, sintered ore, and return slag are often fed in larger sizes, but better results are secured by moderate crushing.

It is also now recognized that thorough mixing of the charge is conducive to the best furnace work. In the old days stress was sometimes placed upon the necessity of feeding the ore, flux, fuel, and slag in layers. With the mechanical feeding now in general use such a practice would be impossible, even were it desirable. The better the mixing the more rapid and efficient will be the smelting. This principle has never been questioned in making a crucible fusion in the assay furnace, and the blast furnace is simply a crucible on a large scale. The reactions taking place in it are practically the same as those occurring in the crucible. In the blast furnace we attempt to correct the disturbing effect of the moving charge column and the upward rush of the blast by adopting the methods of preparation which have been described.

The use of sintered ore in the lead blast furnace has proved so successful that at least one plant, that of the International Smelting & Refining Co., at Tooele, Utah, has adopted the plan of sintering a large proportion of the total blast-furnace charge. The tonnage of the blast furnaces is high, but little information as to the metal recoveries has been published. Dwight-Lloyd sintering machines are used. These machines undoubtedly represent the most advanced type of sintering apparatus. It has been urged against them that they are extremely sensitive. That is true; but it is also true of the blast furnace. In both cases the work is continuous and there must be a constant adjustment of the conditions necessary to good results. In the sintering process the sulphur must be maintained within rather narrow limits. In addition, the crushing, mixing, moistening, and feeding of the charge demand close attention. The

working out of the mechanical details in connection with the process has been a difficult problem, and no doubt further improvements will be made. Most of the inferior results ascribed to the Dwight-Lloyd process can be attributed to defective supervision and to irregularities in the feed. To offset the troubles caused by frozen ore and concentrates various methods for thawing have been devised. In severe climates this becomes quite a problem, and much remains to be done before it is satisfactorily worked out.

The chemical composition of the slags produced has always been an important point in lead smelting. In recent years, however, the views of metallurgists in general upon the subject have been considerably modified. Experience has shown that many of the irregularities in furnace work, formerly thought to be due to the composition of the slag, are the results of other causes. Laboratory experiments have thrown some light upon the constitution of lead slags. It is now known that these slags, instead of being definite chemical compounds, are in reality mixtures of a number of compounds and oxides, probably existing in a different state of combination while in the molten condition than when solidified. These compounds can exist in a great variety of proportions without materially changing the fusibility or fluidity of the slag. It is recognized, also, that certain oxides, such as those of aluminum and zinc, may not be combined with silica at all, but may be in igneous solution in the slag.

The physical properties of the slag are dependent upon the number and kind of its various components, but, within reasonable limits, there is not so much variation as was formerly supposed. The present-day practice is to place more emphasis upon the questions of temperature and fluidity. No slag, no matter what its composition, will work well if it does not flow freely from the furnace and does not remain liquid long enough to permit of a good separation of the slag and matte. In many cases this is simply a question of heat. With the larger furnaces and the improved methods of preparing the charge now in use hotter slags are produced. This has led to the employment of slags formerly considered to be too refractory for lead-furnace work. The slags still conform generally to the singulo-silicate type, but often there is a considerable proportion of bisilicate. In selecting the proper slag the metallurgist is guided very largely by commercial reasons. In some localities, for instance, where there is a surplus of siliceous ores, he will make a slag high in silica, so as to smelt more of the ore which is most abundant. So also he will generally use as much lime as possible, because it is a cheaper flux than iron. In this connection it may be said that high-lime slags are more in use than they were in times past. The higher temperatures employed and the lower zinc contents of the present-day charges have made this possible. Of course it is generally held, also, that high-

lime slags are beneficial metallurgically. The old idea that lime and iron oxide should exist in the slag in certain definite molecular proportions has been discarded by most metallurgists. This so-called "type theory" was supported for a long time, and was considered by some to be extremely essential. Theory and practice have combined, however, to indicate that the things to be considered in adding lime to a charge are the question of cost, and the percentages of silica, alumina, zinc, and magnesia present, rather than the ratio of the lime to the iron. Ordinarily all the lime possible that will still permit the slag to flow readily will be used.

Lead slags are now regularly made carrying from 35 to 37 per cent. of silica and from 24 to 26 per cent. of lime, including magnesia and barium oxide figured as lime. In these slags the zinc and alumina are generally low. High-silica slags are usually considered as being liable to contain a rather high percentage of lead. To some extent this is true, but such a slag runs well, chills slowly and keeps the furnace in good condition. It is conducive to long campaigns. Furnace accretions are always basic, and high-silica charges discourage their formation.

A discussion of slag composition naturally leads to a consideration of the subject of metal losses. With the introduction of bag houses, better methods of sampling, and the general use of the wet assay for lead, a much closer check can now be made upon the efficiency of a smelting plant. These refinements have shown that in most cases the metal losses are greater than they were formerly supposed to be. They have led the metallurgist, also, to modify his views somewhat as to what constitutes a good, economical slag. Twenty years ago it was usually the aim of the lead metallurgists to produce slags and mattes containing the lowest possible percentages of the valuable metals. Comparatively little attention was paid to any other feature. The evils of over-reduction were recognized by some, but not very clearly. The character of the work done was judged mainly by the slag and matte assays. About ten years ago a large smelting plant in Colorado was noted for the extremely clean slags which it was producing. The average wet lead content of the waste slag for one year was 0.67 per cent. The mattes also were very low in lead. The metallurgist congratulated himself upon the figures. There was no bag house, but careful determinations of the lead smelted and recovered for the period showed that the lead losses really were very high. This was confirmed later by analyses of the flue gases. The furnace tonnage was low. At another plant, operating upon the same class of ores, the speed of the furnaces was greatly increased by screening most of the oxidized ores and briquetting the fines. The slags and mattes were unusually clean. A low-silica slag was made, and a considerable production of speiss showed a condition of very strong reduction all the time. Yet the lead losses were extremely high, as shown by the balance sheets at the end

of each month. The work not improving, experiments were begun for the purpose of determining the flue-dust and volatilization losses. The dust at the foot of the blast-furnace stack was found to contain 60 per cent. of lead, and analyses of the gases passing out of the stack showed where the lead was going. The furnaces were low and a high blast pressure was maintained to force the tonnage. Some improvement was effected by lowering the blast pressure and tightening the charge. These illustrations show that the slag and matte assays are not the only points to be considered in lead smelting.

The question of flue dust and fume has always been a difficult one for the lead metallurgist. While bag houses are usually effective, they are expensive to build and to operate. Often they show practically no profit. Collecting, as they do, all of the volatile solids, such as arsenic, cadmium, selenium, and tellurium, they have introduced new problems into the industry. In the latest practice roaster gases are neutralized and filtered also, thus increasing the sulphur and zinc contents of the bag-house fume. The epigram of Anton Eilers, enunciated years ago, that "the best way to handle flue dust is not to make any," still holds good as the ideal toward which the metallurgist should strive. Modern practice has succeeded in reducing the flue-dust and fume losses very materially. This has been the result largely of the changes in methods that have been referred to.

At Midvale, Utah, the blast-furnace bag-house dust formerly carried from 40 to 55 per cent. of lead. This has been cut to an average of under 20 per cent., on a smaller amount of dust. In the meantime the arsenic has risen to about 40 per cent. The arsenic is eliminated almost entirely through the flue as sulphide. Practically no speiss is made and the bullion is remarkably free from arsenic, even though the percentage of that element is high in the charge. As this smelter is equipped with an arsenic plant the large percentage of arsenic in the fume is no disadvantage. Where there are no facilities for collecting and refining arsenic it simply forms one more undesirable product of the bag house.

The formation of arsenical speiss is an evil. It consumes iron, reduces the furnace speed, increases the volatilization of lead, causes trouble in the crucible and settling boxes, and makes an additional by-product which has to be rehandled. It is the accompaniment of low-silica slags, low sulphur in the charge, and powerful reducing action. Its production can be controlled in almost every case.

The question of coke consumption in the lead blast furnace has been the cause of some controversy. The literature of the subject is vague and often misleading. It is usually stated that a certain percentage of coke is used under certain conditions. No information is given as to the chemical and physical characteristics of the coke. Every metallurgist knows how serious are the troubles caused by irregularities in the grade

of the coke. With the charges as now prepared, the main function of the fuel is to produce heat. The reducing action, except in the case of lead carbonates and iron oxides, is largely secondary. But there are other sources of heat in the charge. The oxidation of sulphur and arsenic, the combination of metallic iron with sulphur, arsenic, and oxygen, and the energetic reactions which take place in the formation of the matte and slag all produce heat. The amount of coke to be added, therefore, depends to some extent upon the heat-producing power of the charge itself. In addition, there must be considered the chemical and physical constitution of the coke, the physical condition of the charge, the temperature of the atmosphere, the amount of moisture in the charge and in the air, the condition of the furnace, and the composition of the slag. There must be sufficient fuel to produce a hot, easily running slag. Experiments have shown that considerable lead and silver can be separated from some slags by simply subjecting them to a very high temperature. This causes a still more complete series of reactions between the various oxides, sulphides, and silicates in the slag. With a good charge and with good coke, an amount equivalent to 10 per cent. of fixed carbon, as figured against the weight of the charge (less slag and fuel), may be considered the minimum that can be used. Sometimes less is sufficient. Often it will be necessary to use more, up to 11½ or 12 per cent. In any case the metallurgist must be guided by conditions, and vary the amount of coke as he would lime rock or any other item.

The excessive amount of fine ore formerly smelted led to the practice of wetting the charge before putting it into the blast furnace. As in the sintering processes, moisture renders fine ore porous by preventing it from packing. It keeps the furnace top cool and reduces the amount of flue dust made. In slow running there is always a tendency for the heat to creep up in the furnace. This is counteracted by the water in the charge. Owing to the fineness of the ore particles and the slowness with which they descend, the charge is perfectly dried by the waste heat of the furnace before it reaches the fusion zone. Experience has shown that with fine ore a wet charge can be smelted with less coke than a dry one. With the coarse charges and rapid running which are now the practice there is need for but little water. The pieces of ore and flux are so large and descend so rapidly that if they are saturated with moisture they may not be dried out in time, and thus may chill the furnace. With a coarse charge and fast running, also, there is little tendency for the heat to creep up, and little flue dust is made. Sufficient water should be added, however, to moisten the fines. I have known a heavy rain or snow storm to be followed by cold slags and poor reduction at the blast furnaces. To remedy this condition it is necessary to add more coke.

The volume and the pressure of blast required vary so much that it is hard to give any definite figures. In general it may be stated that

the pressures now used are not so high as they were when tight charges were the rule, but comparatively high pressures are still used to increase the furnace tonnage, when the other conditions are such as to permit it. The air pressure should be kept as constant as possible, and should come from the power house in rapid pulsations. At most of the larger plants the blast pressures used range from 30 to 40 oz. per square inch.

To produce a sufficiently heavy matte fall, coarse iron pyrite, low in zinc, is often added to the charge. This usually raises the slag and matte temperatures and facilitates the elimination of arsenic. As at present conducted there is often considerable pyritic action in the lead blast furnace, with a large production of sulphur dioxide.

Much attention has been devoted to the problem of a proper separation of the matte and slag. Settling furnaces have been devised on the reverberatory principle, using fuel to keep the slag hot and fluid. Most of these have been abandoned. Usually there has been trouble on account of zinc, which gradually accumulates in the furnace. The volatilization losses have been high and there has been also the extra cost of the fuel used. In most cases the slag and matte are separated in the fore hearth. This requires considerable attention and must be changed frequently. It is effective, however, when the furnaces are working properly. Experiments have shown that prolonged settling is unnecessary in the case of hot, properly made slags. It is now known that slags have a slightly solvent action for the sulphides of silver, lead, and copper. Beyond a certain point, therefore, these sulphides cannot be settled out. An earlier recognition of this fact would have saved much time and money.

In the foregoing review of the subject only the more important features in the modern practice of lead smelting have been considered. It is impossible to refer to many important details. In the industry there are many factors, the right adjustment of which is necessary to success. As already intimated, the lead blast furnace is sensitive to adverse conditions. In a smelting plant the ores and fluxes received are continually changing, and the metallurgist finds that he must accommodate himself to a new situation almost daily. Usually there is a shortage or a surplus of some important ingredient of a good charge. At the silver-lead plants there is nearly always a deficiency in the supply of lead, and an excessive amount of sulphides. Often the metallurgist puts a charge on his furnaces that he knows will not give the best results. But he must keep his furnaces going, and he smelts what he has.

As the object of smelting is to make money, the commercial side of the industry must be kept uppermost. Apparently poor metallurgical work may mean the greatest financial success. As between processes, that one will be selected which gives the largest returns in money. At most plants there are local considerations that may modify what is generally held to

be the best practice. In hardly any other metallurgical industry is it so difficult to lay down general rules. But the underlying principles have been worked out and the conditions that are essential to good practice are known. That the results obtained are often not what they should be is due to the fact that these conditions could not be realized.

Discussion

L. S. Austin, Salt Lake City, Utah.—With the union of the Guggenheim interests and the American Smelting & Refining Co. in 1900, competition among the lead-smelting companies appeared to have been largely eliminated. This condition prevailed for but a short time, owing to the erection of smelteries by competing companies. Naturally, as competitors, the older companies endeavored to keep improved methods to themselves, not realizing, it would appear, that such knowledge could never be bottled up, but would drift into the common field of practice. Of late it has begun to be felt that more is gained than lost by adopting a franker and more open policy. The line, however, is drawn in giving details of costs, which, fully known, it is felt, would be taken advantage of by a rival.

As Mr. Palmer says, steady and substantial progress has been made, and latterly, I may add, the improvements have been radical.

Sintering by the Huntington-Heberlein method or with Dwight-Lloyd machines has effected an important saving over reverberatory roasting, the cost being, say, 50 to 75c., as compared with $2 per ton by the older method. Moreover, the large use of a sintered product has resulted in doubling blast-furnace tonnage. However, the change has not been an unmixed blessing, as in the machine roasting little time can be given to the breaking up of the blende, and it has been impossible to decompose largely the zinc sulphide, so detrimental in later blast-furnace treatment. The satisfactory roasting of the blende is still an unsolved problem. Nor is this all. For the best reduction it has been found that a part of the charge should be crude (unsintered) ore, indicating that the already fused material is only too ready to change to slag, thus lessening the opportunity for better reduction.

So far as the mechanical handling of ores is concerned, the importance of the saving to be effected has perhaps been exaggerated. To be sure such mechanical handling has been reduced to a low figure, but even 15 years ago it was cheaply done. Philip Argall gave the cost of unloading, sampling, and coarse crushing (the ore being delivered in the bins) at the mill of the Metallic Extraction Co., Florence, Colo., at 11c. per ton. Cutting such costs in two, or to 5½c. per ton, makes no important saving, especially when we consider the interest on the greater cost of a present-day installation.

Referring to the boshing of the furnace, the introduction of air, the height of ore column, and the feeding, interrelated factors as they are, I would say:

It is a matter of observation for 30 years or more that after a furnace has arrived at a certain shape during its campaign, owing to shaft accretions, it will drive faster. With no extended flue system or bag house, as in earlier practice, it was necessary that the furnace should be expanded at the top of the ore column in order to prevent the formation of excessive flue dust, and for the same reason the furnace was run slowly. To-day, with the large tonnages put through, this is no longer possible. The air rises through the charge at an estimated velocity of 22 ft. per second, as against 5 ft. per second in early practice. But with the removal of the fines, and with the aid of the flue system and the bag house, such rapid running is now quite possible.

The notion that the shape, size, and direction in which the tuyères are pointed are of importance, seems to me absurd. Of course such openings through the jackets should be ample, but, that penetration is in any way modified, conforms neither to reason nor to practice. The air enters the furnace and rises through the charge by the path of the least resistance, often an intricate and winding way. A partly slagged-over tuyère takes the air, not straight forward into the furnace, but through cracks, openings, and cavities and so to the more open part of the charge. This air is not to be recognized as a gust of wind traversing a free space, but rather as air pent up and seeking to escape where it can most easily pass through.

As respects the height of column, the older notion was "to increase the ore column in order to give more time for reduction." That idea we must give up. Furnace gases, in a well-working furnace, carry no more than 5 per cent. of CO. We know that this gas so diluted has but little reducing power. Moreover, it has less than a second in which to perform reduction even with a high column. Actually, reduction must be due to the action of the glowing coke in the melting zone.

I have already referred to the path of the air by channels of the least resistance. Hence the importance of suitable mechanical feed, supplemented by trimming the furnace by hand, so that the rising air escapes uniformly from the surface of the charge.

The matter of slag composition was threshed out years ago, and the literature of the subject is extensive and conclusive. Clean slags were made then and furnacing was done with as much skill as to-day. In some of the discussions the chemical language of "oxygen ratio and of single and bisilicate types" was used, but practically, as now, percentage composition was carried in mind. We speak to this day of a whole, a three-quarter, etc., slag, but it is only a convenient way of designating a type. At no time in the past generation did we think that a slag was other

than a complex silicate with alumina and zinc sulphide not a constituent of that silicate, but dissolved in it. Quite in the past, as is also now the case, only fuel enough was used to give a hot slag of the proper fluidity. With our high tonnages the exothermic reactions of slag formation take place more quickly and so more energetically, and hence the lower percentage of coke that can be used. We have the disadvantage, too, owing to the use of mechanical charging, that the metallurgist cannot, as in the hand-charging days, closely see and study the phenomena of feeding, the action of blowholes, the appearance of the surface of the charge and the escape of the rising gases. Theory has had to replace in part actual observation.

In this, the Utah smelting center, the complaint is that, as the oxidized surface ores have been replaced by sulphides (especially blende), the difficulties of smelting have increased. To take care of this zinc sulphide, a more irony, less limy, and consequently less siliceous, slag has been made. The high-lime slags are better suited to ores low in zinc. The disadvantage of the more basic slags is, however, that, due to the lessened difference between them and the matte, separation is less complete. This brings into consideration the question of the use of the settling furnace; because of high-zinc slags, it has been tried and rejected in numerous instances.

It is the practice to send back to the blast furnace slag shells and other foul slag, with the idea that such additions aid in the smelting operation —viz., that slagged material would promote the formation of new slag, while incidentally its contained silver and lead would be largely recovered. Slag additions displace new charge, and it now appears that it will pay better to waste all the slag. We have to remember that it is now high in zinc and in sulphur, both elements we are glad to be rid of.

It is a fact that may be forgotten that when the slag is running cold, analysis shows that the iron content is low, and the metallurgist proceeds to increase it, when, were he to increase fuel, he would find that the iron would rise to the required figure.

Were I to predict in what direction we may hope for radical improvement in lead smelting, I would say that it would be in the use of the reverberatory furnace, principally because of the saving in fuel, or a reduction to 60c. per ton of materials smelted, against 90c. per ton as now prevailing. One of the first objections to suggest itself would be the difficulty of preventing the hearth from coming up and the leakage of lead into the foundation. However, in refining both lead and copper such hearths can be maintained. The next point would be, that should escaping gases be filtered in a bag house there would be their high temperature to overcome. However, since in passing the waste-heat boiler that temperature has been largely reduced, the necessary further cooling should not be difficult. Moreover, we have the Cottrell system as another

method to fall back upon. In operation coal would have to be added to the charge and no doubt the fire carried with a neutral or reducing flame. Reverberatory smelting has the advantage that to correct the charge carbonaceous material, fluxes, oxidized ores, or roasted material could be added at any time, and that the charge need be tapped only when ready. There would be no tap-hole difficulties, no slagged tuyères, no hearth and shaft accretions to interfere. The molten contents of the furnace could be inspected and tested. There would be little or no flue dust, though no doubt considerable fume would have to be caught.

L. D. ANDERSON, Midvale, Utah (communication to the Secretary*).—Some further light has recently been obtained on certain points mentioned in Mr. Palmer's paper. With respect to the angle of the bosh in the furnace lines, a rebuilt furnace of only about 10° or 11° has been running beside furnaces of the old angle of over 20°. As far as observed, the new furnace seems to run much hotter and obtain a larger percentage of lead contents as direct bullion output. The matte fall, however, is considerably smaller, and it is necessary to carry more sulphur on the charge in order to maintain this fall high enough to insure clean slags. Judging from slag analysis and percentage of matte fall, the narrower furnace burns off half, and even more, of the sulphur as SO_2 as compared to only a third, or less, for the wider furnace. In these instances both furnaces are 48 in. at the tuyères, the difference in width being in the stack. The narrower furnace would appear to be superior to the wider one in that it has less tendency to form awkward hanging crusts, its crucible stays hotter, and, considerable heat being supplied by the combustion of the sulphur, a noticeable saving in fuel consumption can be effected. These remarks apply to rather coarse charges; for fine charges the differences would not be so marked.

I feel that the matter of the correct furnace shape is yet far from being settled. Every metallurgist has undoubtedly noticed that often after a furnace has been in blast for some time its work improves and the slags become surprisingly clean. Furnaces have been observed with crusts in them so heavy as to make it seem almost imperative to take them out of blast, whose slags nevertheless were so clean as to make the metallurgist reluctant to disturb them, until finally blowing out becomes imperative on account of the reduced tonnage. Experiences of this kind lead to speculations as to whether better reductions might not be obtained by making the shaft walls converging instead of diverging, somewhat after the lines of the iron blast furnace. In other words, why not make the furnace of the ideal shape at the outset, instead of waiting for it to form its own "natural" shape and running all the chances of irregularity in the formation of the crusts?

* Received July 31, 1914.

In this matter of furnace shape there also comes in the question of the amount of lead which it is reasonable to permit in the flue dust. An old, heavily crusted furnace tends to "blow through" and to produce an excessive amount of flue dust, high in lead. It has been found that, other things being equal, the percentage of lead in the flue dust or the bag-house dust is a good index of the work accomplished in smelting as a whole. Theoretically the bag house catches all the metals escaping from the blast-furnace tops. In practice, however, it does not actually save them entirely. Bag-house and flue dust have to be rehandled, and in the course of the rehandling several losses occur. Careful observation shows that when the lead in the various dusts is maintained at the minimum the percentage of recovery will reach its maximum. For this reason daily assays of the bag-house dust and adjustment of furnace conditions so as to maintain a low lead content in the dusts will prove of great value in the problem of keeping the metal losses at a minimum. The experiences leading to these conclusions appear to corroborate Mr. Palmer's remarks on the high lead losses experienced at plants whose slags were most admirable for cleanliness, but which had apparently not given full attention to the matter of volatilization losses.

Mr. Palmer's remarks on the desirability of a freer exchange of blast-furnace experiences are warmly seconded. Independent smelters are freely visited by officials of the larger corporation. The industry as a whole suffers when a policy of uncommunicativeness prevails. The handling of the blast furnace, after all is said and done, becomes finally a matter of personal judgment. Exchange of ideas on details can therefore have scarcely any effect on competition, while it will promote the tone of the entire profession.

Lead Smelting at East Helena

BY EDGAR L. NEWHOUSE, JR., SALT LAKE CITY, UTAH

(Salt Lake Meeting, August, 1914)

THE lead smeltery at East Helena, Mont., controlled by the American Smelting & Refining Co. since 1899, has been in continuous operation for the past 25 years. Most of the old smelting and roasting practices have been superseded by newer and more modern methods, but on account of the difficulties involved in changing the original construction, some of the methods in use of handling materials are not the most modern. This plant, however, is particularly interesting as being one of the few essentially lead smelteries in the United States. At Federal, Ill., Collinsville, Herculaneum, and possibly other places, similar work is being done, but it is doubtful whether any of them produces as much lead as the East Helena plant, whose monthly production averages 6,500 tons.

The situation, 5 miles east of Helena, is convenient to both the Great Northern and Northern Pacific railroads, while the electric power is obtained from Canyon Ferry, 12 miles east on the Missouri river.

Local quarries supply the requisite amount of lime rock, which the contracts require to be crushed to 5 in.

Of the 450 men on the monthly pay roll, a census showed 200 Austrians, 110 Americans, and 30 Italians, the balance being made up of miscellaneous nationalities.

The ore supply is mainly derived from the Cœur d'Alêne mines in Idaho, and consists of crude ore, coarse and fine concentrates, middlings, and slimes. About one-half of the total output of the Cœur d'Alêne camp is sent to East Helena. An average analysis of these ores gives approximately 40 per cent. lead and 11 per cent. sulphur.

Local shipments, varying from month to month, are mostly highly siliceous gold and silver ores carrying little lead. Small shippers, leasers of old mines, and prospectors are very numerous throughout Montana, and it is the policy of the company to encourage these men by every possible means, including the free sampling and analysis of their ores and aid in the development of their mines and prospects, whenever their results show promise.

Sampling, Crushing, and Bedding

In order to obtain a desirable roasted and sintered product for the blast furnaces, and as complete an elimination of sulphur as possible, practically all the ore is carefully bedded in bins of from 1,200 to 1,500 tons capacity. The accurate bedding of the ores is considered as essential to successful metallurgical practice and hence the utmost stress is laid upon this point, the importance of which is clearly demonstrated, particularly in the winter months, when during days of 5° to 20° below zero all the ore freezes both in the cars and in the bins, the bedding is rendered inaccurate, and the result is evident in the subsequent erratic behavior of the furnaces. The slimes and fine concentrates are bedded direct and the crude ore and coarse concentrates are unloaded from the box cars into a pan conveyor, which discharges into the sampling and crushing mill. This mill has a capacity of 45 tons per hour, crushing to 0.41 in., and taking a mechanical sample of $\frac{1}{6}\frac{1}{2}\frac{1}{5}$ of the original volume by means of five Vezin samplers. The sample is then cut down by coning and quartering and treated in the usual way.

Rich gold and silver ores are put through a smaller sampling and crushing mill, a $\frac{1}{2}\frac{1}{5}$ part being taken for the sample. These ores are binned separately and go direct to the blast furnaces on account of the high metal losses that would result in roasting. Single car lots are put through a third mill, which crushes to $\frac{3}{8}$ in., taking $\frac{1}{2}\frac{1}{5}$ out for the sample. Twenty-five per cent. of the incoming ore is handled in the two small mills.

Montana mining laws require that $\frac{1}{4}\frac{1}{0}$ of all ore shipped from Montana mines be held by the smelting companies for 30 days pending settlement.

The bins are divided into four divisions, three of the divisions containing the charges for the Dwight-Lloyd sintering machines, the Godfrey preroasting furnaces, and the Huntington-Heberlein converter pots, and the fourth being miscellaneous bins for flux, sintered product, and oxidized ores.

Roasting

In 1907 a small straight-line Dwight-Lloyd machine, 30 by 150 in., was installed for experimental purposes. The results were so satisfactory that in 1908 four additional machines, 42 by 264 in., were ordered. These machines have given great satisfaction and have been a decided success in every way. Until 1912 crushed siliceous ore or barren lime rock was used as a grate dressing, mainly to prevent the charge from sticking to the grates. This practice was discontinued, however, as it was found that by using a straight-slot grate in place of the original herringbone pattern, by regulating the physical make up of the bin, and by keeping the grates open, the grate dressing could be dispensed with. The change from the

herringbone to the straight-slot grate made possible the installation of a grate-cleaning device on each machine, consisting of a steel disk roller running parallel to the direction of the pallet. The charge mixture is trammed from the bins and conveyed to a Robins automatic tripper by means of a belt conveyor and bucket elevator. The tripper feeds the four hoppers above the machine uniformly. A thorough mixture of the charge is made in loading the bedded ore from the bins into the tram cars and also in distributing to the machine hoppers. The moisture, approximately 5 per cent., is regulated to suit the physical and chemical condition of the charge. Ignition is secured by means of a gasoline burner under 80 lb. pressure and the suction varies between 5 and 8 in. of water, depending on the tightness of the charge. The charge which can be put upon the Dwight-Lloyd machine is extremely flexible; varying percentages of matte, flue dust, slimes, and very fine concentrates, mixed with milled ore, crushed to 0.41 in., are daily sintered successfully. The sinter produced is hard and porous, with very little fines. Tonnages vary from 90 to 120 tons per machine day of 24 hr. The speed of the machine is from 16 to 20 in. per minute, and the sulphur is reduced from 12 per cent. to 3½ to 5 per cent. The following is a typical charge made up of approximately 75 per cent. milled ore, crushed to ¼-in. mesh, 20 per cent. slimes, and 5 per cent. flue dust. This charge ran at the rate of 110 tons to the machine, producing an excellent product for the blast furnaces:

	SiO_2	Fe	S	Zn	CaO	Pb	Cu
Before roasting.....	15.0	15.2	11.6	4.1	2.6	39.0	0.1
After roasting.....	16.8	17.2	4.5	4.0	2.9	41.7	0.4

In September, 1912, experiments were begun to determine the advisability of filtering the gases from the Dwight-Lloyd machine. A small experimental bag house was erected containing both cotton and woolen bags. At the end of six months, tensile-strength tests showed the bags to be uninjured by the gases. These tests were continued, with the result that in September, 1913, the gases were turned into the main bag house along with the blast-furnace gases. On account of the high lead content of the charge and the heat developed, sufficient lead oxide is formed, together with a small amount of zinc oxide, to neutralize the SO_2 in the gases.

The blast-furnace matte is preroasted in the four Godfrey furnaces, each having a capacity of 27 tons per day, converting the sulphur from 20 to 10 per cent. This preroasted matte is then mixed with the direct converting mixture containing not over 10 per cent. sulphur. The combined mixture is then blown at a pressure of from 8 to 12 oz. in the Huntington-Heberlein pots, having a capacity of 12 tons each. At the end of 12 hr. a pot is raised by means of a 20-ton crane, the non-agglomerated fines are raked off, and the pot is carried to the breaking floor, where the button is dropped. The large lumps are picked up by the crane and

dropped again, while the smaller lumps are fed into a Blake crusher set to 3 in. and hoisted into bins of 200 tons capacity, from which the product is taken to the furnaces. The sintered product is harder and less porous than the Dwight-Lloyd sinter, with perhaps 60 per cent. of it slagged. Taken as a whole the Dwight-Lloyd sinter is more satisfactory for the blast furnaces than the Huntington-Heberlein sinter. The standard tonnage for the department, consisting of the four Godfrey furnaces and 12 Huntington-Heberlein pots, is 300 tons per day. Following are analyses of the matte before roasting, the direct converting mixture, and the final Huntington-Heberlein sinter.

	SiO_2	Fe	CaO	S	Pb	Zn	Cu
Matte.....	2.0	42.8	0.4	21.0	14.0	5.4	6.6
D. C. mixture.....	11.4	14.7	2.5	9.4	42.0	3.5	Tr.
H.-H. product.....	9.1	27.9	1.9	5.0	35.0	4.7	2.7

Smelting Operations

There are four blast furnaces, 48 by 136 in. at the tuyères, with an effective height of charge column of 18 ft. Each furnace is provided with 16 tuyères, 3½ in. in diameter, supplied with blast at 33 oz. pressure. Three of the furnaces are generally in operation with the fourth held in reserve. At present, however, the four furnaces are running. This necessitated starting up three old hand reverberatories pending the installation of two new Dwight-Lloyd machines under construction. The standard tonnage for each furnace is 220 tons per furnace day, but this has been exceeded by 90 tons. The 4-ton charge car is raised to the feed floor by means of an incline, the hoistmen raising and dumping the car. The lead from the furnaces is tapped into 1-ton pots and sent to the drossing plant, and the slag and matte are run into a brick-lined floor hearth, from which the matte is tapped into Kilker mold cars, from which it is dumped into a railroad car and sent to the mill to be crushed and put into the Godfrey roasting bins. The slag overflows from the settler into a settling pot, where any matte that may have come over settles out, and from this pot it overflows again into a second slag pot. When the settling pot is full it contains a large percentage of matte and hence is kept separated from the slag shells that are returned from the slag dump. These shells are loaded from the dump into railroad cars by a locomotive crane and go direct to the furnaces. The matte from the settler pots is returned to the mill and crushed for the roasting bins. The following is an extract from a daily report:

Slag Analysis, Per Cent.

Pb.	Ag.	SiO^2	FeO	MnO	CaO	MgO	Zn.
0.90	0.30	33.2	31.9	2.6	18.4	2.1	5.0

Charge, Per Cent.

Pb.	S.	Coke	F.C.	H.-H	D.-L.	Slag Resmelted
31.9	3.8	12.92	10.1	36.5	52.6	15.4

Matte fall, 14.1 Average blast, 33 oz. Average tonnage of three furnaces, 260

The matte on this day contained 18.7 oz. silver, 13.4 per cent. lead, and 9.7 per cent. copper. The specific gravity of the slag was determined to be 3.3 and that of the matte 4.2.

The report speaks for itself, but there are several points of special interest: (1) The high percentage of lead on charge, with the accompanying small amount in the slag; (2) the high speed combined with good reduction; and (3) the fact that there was 89 per cent. roast on the charge. Such a large percentage of roast is handled at few plants.

The Eilers-type slags are carefully adhered to, experience at this plant having taught that in so doing the best results on the furnaces are obtained.

The so-called lead smelters of Colorado and Utah run with from 8 to 15 per cent. lead on charge. With them the lead is simply a means of collecting the precious metals and saving the small amount of lead in their ores, whereas at East Helena, on the other hand, with from 30 to 40 per cent. lead on charge, this order is reversed on account of the high percentage of lead in the ores, and their low silver and gold content. As the percentage of lead on the charge increases, the problems of reduction and good blast-furnace work become greater and the furnaces become extremely sensitive to changes that would hardly affect them when running on a lower lead charge.

Whereas 5 to 6 per cent. Zn in the slag, produced from a low-lead charge, gives little trouble, its importance is greatly increased when we have from 30 to 40 per cent. lead on charge. Smelting a high-lead charge is a distinct problem, occupying a field of its own, whose difficulties offer no comparison to blast furnaces running on a low-lead charge.

Three brick bag houses, 43 by 133 ft. and 50 ft. high, each containing 1,260 cotton bags 18 in. in diameter and 30 ft. long, filter the blast-furnace and Dwight-Lloyd machine fumes. The bag houses are connected to the furnaces and Dwight-Lloyd machines by flues 1,570 ft. long. The suction in the flue is obtained by means of a fan making 136 rev. per minute and driven by a 100-h.p. motor. The temperature of the gases entering the houses varies from 36° in the winter to 92° in the summer. The average fan pressure on the bags is 0.40 in. water and the suction in the flue 0.70 in. The gases entering these bag houses contain a very small amount of SO_3 and in consequence the life of the bags is very long, approximately four and a half years.

The collected dust is burned, for the main purpose of getting it into such a state that it can be easily handled on the roasting bins. Formerly, when high arsenical ores were treated at this plant, the raw dust contained 40 to 50 per cent. arsenic, which on burning was reduced to 15 per cent. The following is an analysis of the bag-house dust before and after burning.

	SiO_2	Fe	Mn	Bi	Zn	CaO	MgO	S	As	Sb	Cu	Pb
Raw dust.....	2.6	0.7	Tr	Tr	2.4	0	0	10.2	1.2	Tr	0	68.0
Burned dust....	3.2	1.3	Tr	Tr	1.8	0	0	7.8	1.6	0.3	0	62.8

The lead bullion is drawn from the furnaces into 1-ton pots. These are trammed into the drossing plant, adjacent to the furnace building, hoisted to the level of the drossing pots and dumped by means of a small auxiliary hoist. There are four large pots having a capacity of 54 tons each. While being filled the pot is heated, so that when full it is only necessary to allow it to stand a short time before removing the dross by skimming. This is done by hand, and generally requires only a short time. The lead is tapped from the bottom of the pot into bars weighing 90 lb. each. These are loaded by hand and shipped to the refineries at Omaha and Chicago, while the dross is put directly back into the blast furnaces. The monthly production of lead bullion is approximately 6,000 tons on a three-furnace basis.

The greatest stress is laid upon the safety and welfare of the men at the East Helena plant. Every possible precaution is taken to insure the men against injury and to promote their general welfare. A safety inspector is in charge of this work and every effort is being made to obtain the co-operation of the men and their foremen with the staff in charge to secure a high degree of efficiency in this department.

Discussion

G. C. Riddell, East Helena, Mont.—As nearly as I can recall at the present moment, there are only four or five large plants on the continent where the lead on charge is carried in the neighborhood of 40 per cent. These are at Herculaneum, Collinsville, and Federal, Ill.; Trail, B. C.; and East Helena, Mont. Our object at the latter point is to smelt a maximum amount of Cœur d'Alêne lead sulphides in as compact and self-fluxing a charge as possible, and to this end we are now smelting a mixture that at times carries up to 42 per cent. lead.

The monotony of producing the 7,000 tons of bullion shipped each month to the Omaha and Chicago refineries is relieved somewhat by the fact that the 15,000 tons monthly of Cœur d'Alêne ores that are coming to us carry enough zinc to provide the one-half slag used with from 6.5 to 7.5 per cent. zinc day after day. During the month of March, 1914, three lead furnaces in continuous operation, each 136 in. long, averaged 249 tons per furnace day, exclusive of coke and slag shells, which is smelting at the rate of 5.5 tons per square foot of tuyère area. The R. P. M. of the metallurgist himself on this 40 per cent. lead-7 per cent. zinc operation are not a matter of record, but can perhaps be imagined.

The presence of so much lead on charge results in a very light slag fall, and it is this matter of the small amount of slag produced per ton of charge that makes the local metallurgy an interesting proposition at times. One ton of charge makes less than one-half ton of slag, comparing with

three-quarters of a ton at most plants, and with the high lead burden carried in the furnace charge there is a great tendency for the metals to crowd into the slag produced. Needless to say, the lead is kept in the neighborhood of 1 per cent. (wet), in the slag, only by the most careful refinement and control in preparing and handling the charge at roasters and feed floor, distributing the feed at the furnace top, manipulating the furnace itself, and by attention to the type and analysis of the slag.

While the slag fall is abnormally light, the matte fall in this operation is exceedingly heavy. The Cœur d'Alêne ores that form the basis of our operation carry in the neighborhood of 10 to 13 per cent. sulphur, and 90 per cent. of the smelting stock is composed of these roasting ores. No other one item must be so carefully watched for its effect on the blast-furnace metallurgy as the matter of preparing in the roasting department a hard, firm, satisfactorily sintered roasted product, low in sulphur.

We have under way at East Helena, at the present time, a very interesting and promising series of roasting experiments, designed to cut the matte fall of the past few years, 15 to 16 per cent., down to 7 or 8 per cent. The blast roasting of these zincky lead ores has hitherto left a residual sulphur of 4 to 5 per cent. in the sintered product, but at the close of a long series of experiments on both the H. and H. and the D. and L. processes we have recently hit upon a combination of the D. and L. and the H. and H. which brings this final sulphur down to from 2 to 2.5 per cent. A trial run with this combination process on 100 tons of charge was entirely successful from the roasting point of view and we are now making a 2,000-ton test, to check up the first run and to demonstrate the behavior of this sintered product on the blast furnace itself. If the test comes up to expectations, and to the preliminary run, a detailed description of the method will perhaps be of interest to those members directly concerned with the smelting of high-lead charges, and will be forthcoming later.

R. C. CANBY, Wallingford Conn.—Mr. Riddell says they adhere to the Eilers-type slag on the charge, placing emphasis on that. I would like to ask how closely they adhere to that type of slag?

MR. RIDDELL.—We still do that, at the East Helena plant, although in doing so we do not feel that it is a necessity. As we change over from one smelting campaign to another, we generally move from one well-defined slag type to another. That is to say, if the two-fifth type at any time becomes too irony for the ore situation, we go to a 0.444, one-half, five-eighth, or a three-fourth slag. The 0.444 is not an Eilers type, by the way, but we feel in general that the closer we stick to a definite type slag, the longer does the furnace shaft remain free from accretions, and the better does the crucible behave. While we recognize the Eilers types as good slags, we feel that we have found a number of other types that can be used to advantage when the occasion requires.

The International Lead Refining Plant

BY G. P. HULST, EAST CHICAGO, IND.

(Salt Lake Meeting, August, 1914)

THE Parkes process lead refinery of the International Lead Refining Co., at East Chicago, was built by the International Smelting & Refining Co. to treat the lead bullion produced by its Tooele plant, at Tooele, Utah. The plant was designed and constructed under the direction of the writer. Ground was broken Apr. 16, 1912, and exactly six months later the plant was in operation and lead was being cast. The capacity of the plant,

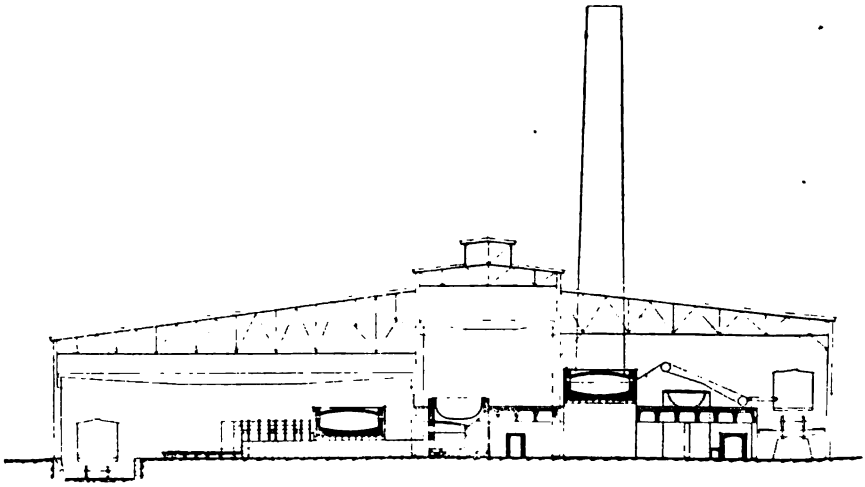


FIG. 1.—CROSS-SECTION THROUGH MAIN BATTERY.

running 25 days per month, is 5,000 tons. The four products of the plant are common lead, corroding lead, antimonial lead, and Doré bullion.

The site is an area, 64 acres in extent, lying north of 151st Street, between McCook Avenue and the Canal. The Indiana Harbor Belt Railroad, Baltimore & Ohio Chicago Terminal, and Pennsylvania Railroad enter the yard at the plant. There are five storage tracks and space for storing 100 cars in the yard. The main lead to the storage tracks passes over a 100-ton track scale on entering the yard.

The plant is so arranged that all the principal operations are performed in one main building. This is a steel and concrete structure, 480 by 180 by 50 ft., whose various levels are shown in section in Fig. 1. Three broad-gauge railway tracks enter the building, and two 15-ton electric cranes, one 28 ft. and one 77 ft. span, travel the entire length. These cranes serve all departments and greatly facilitate the handling of material.

The main battery consists of the following apparatus:

	Capacity Tons
Two sample kettles	45
Two liquating kettles	60
Two softening furnaces (inside dimensions, 13 ft. 6 in. by 28 ft. 2 in. by 31 ft. 2 in. deep)	300
Two desilverizing kettles	100
One refining furnace	300
One molding furnace	200

The arrangement is such that the lead flows by gravity from one piece of apparatus to the next and is finally hand-molded and loaded by trucks into cars.

For the treatment of by-products resulting from the main refining operations, the following equipment is provided:

	Capacity
Three residue furnaces, each 8 by 16 ft. by 20 in. deep (inside measurement)	30 tons each
Two circular blast furnaces, each 42 in. in diameter at tuyères by 14 ft. high, with five 3-in. tuyères	40 tons each
Eight retort furnaces	1,300 lb. each
Two tilting cupels, Rhodes type	5 tons each

Common lead is further refined to yield a product suitable for corroding by the Hulst improved crystallizing process. The equipment of this department consists of the following:

	Capacity
Two crystallizing kettles	60 tons
Four heating kettles	20 tons
One press.	

Gases from the cupel, residue, and blast furnaces are conducted through brick and steel flues to a single bag house. The bag house is a building of brick and steel, 50 by 65 by 50 ft. The interior is divided into four separate chambers, each containing 144 woolen bags, 18 in. in diameter and 30 ft. long. The bags are shaken by an electric-driven automatic shaking device. The gases are delivered to the bag house by an 8-ft. American Blower Co. fan, driven by a 35-h.p. motor.

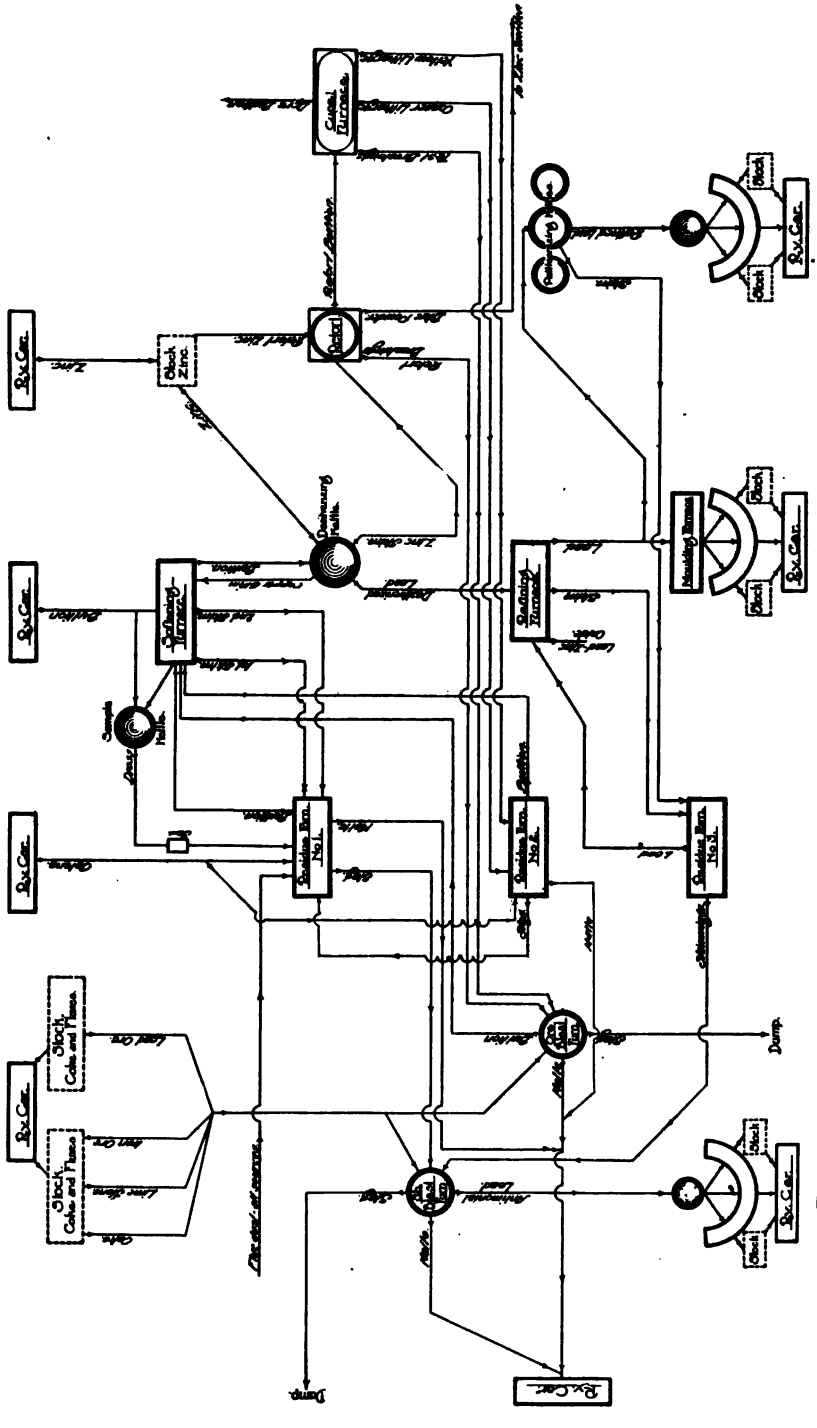


FIG. 2.—FLOW SHEET OF THE EAST CHICAGO PLANT OF THE INTERNATIONAL LEAD REFINING CO.

Steam, for use in the softening and refining furnaces, and for operating the air compressor, is furnished by two 90-h.p. Hawkes boilers. Electric power for all operations is purchased from the Northern Indiana Gas & Electric Co. Water is obtained from the city mains, which supply a 50,000-gal. tank elevated 50 ft. Waste water is returned to a 100,000-gal. sump tank and is pumped to the elevated tank.

The change house is a brick building, 35 by 85 ft. It is equipped with sanitary toilets, wash basins and lockers. One room is arranged as a lunch room for the men. The offices and laboratory are housed in a brick building 36 by 128 ft.

The accompanying flow sheet, Fig. 2, shows diagrammatically the handling of raw materials and by-products.

FIG. 3.—DESILVERIZING KETTLES.

Lead bullion from Tooele containing about 99.5 per cent. lead, 65 oz. silver, 0.4 oz. gold, and varying amounts of copper, antimony, arsenic, zinc, and bismuth, is received at the refinery in sealed cars and after being weighed is delivered into the softening furnaces by means of a steam-driven conveyor, constructed by Howe Scale Co. The sides and ends of these softening furnaces are water jacketed from the base plate to 3 in. above the slag line.

The products of the softening furnace are first skims, second skims, and softened bullion. The first two are sent to residue furnace No. 1. The bullion flows to the desilverizing kettles. The copper skimmings are charged into the softener.

In the desilverizing kettle bullion is treated with zinc and skimmed, yielding zinc skims and desilverized lead. Fig. 3 is a view of the desilverizing kettles. In the kettle in the foreground the last skim is being taken and skimmings transferred to a large mold. In the background the crane is seen handling a large skim block.

The zinc skims go to the retorts and the desilverized lead to the refining furnace. The products of the refining furnace are lead-zinc oxide, refinery skim, and refined lead. The first product, depending on its composition, is treated in one of the residue furnaces or in the blast furnace. The skimmings go to residue furnace No. 3. The refined lead, in part, goes to the Pattisonizing kettles for further treatment, and the remainder passes to the molding furnace. Fig. 4 is a view of the molding

FIG. 4.—VIEW OF MOLDING FLOOR.

floor and shows the method of handling the bars of lead with the crane.

At the molding furnace the lead is siphoned into molds arranged in the arc of a circle, as shown. The bars are removed from the molds and stacked by hand and are placed in stock ready for shipment.

In the treatment of by-products, the zinc skim produced at the desilverizing kettles is treated in four tilting retort furnaces using oil as fuel. The products of this operation are retort zinc, retort breakings, blue powder, and retort bullion. The retort zinc is returned to stock to be used again at the desilverizing kettles. Retort breakings are sent to the ore blast furnace. Blue powder is shipped to zinc smelters for treatment.

The retort bullion passes to two Rhodes type cupel furnaces. These produce test breakings, copper litharge, yellow litharge, and Doré bullion.

The test breakings go to the ore blast furnace, the two litharge products go to residue furnace No. 2, and the Doré bullion is molded into anodes and shipped to the Raritan Copper Works for refining.

The Pattisonizing kettles, equipped for the Hulst crystallizing process, are shown in Fig. 5.

The lead received at the kettles from the refining furnace contains from 0.08 to 0.12 per cent. Bi. One crystallization reduces the bismuth from 0.08 to 0.05 per cent. and less. For lead containing 0.12 per cent.

FIG. 5.—PATTISONIZING KETTLES.

Bi two crystallizations are necessary, if the crystals are drained by gravity; one is sufficient if crystals are pressed.

With such a low-grade product subsequent treatment for the recovery of bismuth is not profitable. This department will produce 150 tons of refined corroding lead per day, with a bismuth content of 0.05 per cent. or less.

Residue furnace No. 1 receives sample-kettle dross, softening-furnace skims, flue dust from all sources, and galena ore (80 per cent. Pb). The charge is weighed in over a small charging scale, and is so proportioned as to yield products of fairly constant composition. The products are bullion, refinery matte, and antimonial slag. The bullion is returned to the softening furnace. The matte is shipped to the smelter for treatment and the antimonial slag goes to the antimonial blast furnace.

Residue furnace No. 2 receives only the litharge products from the

cupel furnaces and galena. Its products are bullion, slag, and refinery matte. The bullion is returned to the softening furnace, the slag goes to the No. 1 residue, and the refinery matte is shipped to the smelter.

Residue furnace No. 3 treats the skimmings and dross from the refining furnace and from the Pattisonizing kettles. These yield lead and skimmings. The former goes to the refining furnace, the latter to the antimonial blast furnace.

The ore blast-furnace charge is made up of lead ores, coke and fluxes, and the following by-products: Retort breakings, test breakings, and slag from residue furnace No. 3. The three products are slag, matte, and bullion. The slag goes to the dump; the matte is shipped to the smelter, and the bullion is returned to the softening furnace.

The antimonial blast-furnace charge consists of ore, coke and fluxes, and, in addition, antimonial slag from residue furnace No. 1 and skimmings from residue furnace No. 3. The charge weighs 1,450 lb. The coke used is 12 per cent. of the weight of the charge. The products are slag, matte, and antimonial lead. If no ore is used the charge contains no sulphur and no matte or speiss is formed. The charge is carried low in the furnace under a light blast pressure (5 to 6 oz.). Arsenic is burned off and recovered in the flues and bag house.

The slag produced is sent to the dump. It has the following analysis:

	Per Cent.
SiO_2	24.0 to 26.0
Al_2O_3	10.0
FeO	36.0 to 38.0
CaO	10.0 to 12.0
ZnO	12.0 to 14.0
Pb	1.5 to 2.0

Matte, when produced, is shipped to the smelter.

The antimonial lead is run to a liquating kettle, from which it is cast into bars for shipment. Consumers of this product commonly specify that the lead should contain: Antimony, 15 to 18 per cent.; arsenic, less than 1 per cent.; copper, less than 0.5 per cent. This department has been remarkably successful in producing lead conforming to the specifications of the mixed-metal plants.

Acknowledgments.—A description of this plant would not be complete without an acknowledgment of the assistance rendered by R. Ruetschi, in the preparation of plans; C. W. Wilson, L. Crook, and F. E. Stolte, in the work of construction; and Herman Witteborg, in the operation of the plant.

Discussion

L. S. Austin, Salt Lake City, Utah.—One question that I would like to ask is in regard to the sampling kettle which stands beside the softening furnace. How is the sample taken or melted?

G. P. HULST.—The sampling kettles in question are of 45 tons capacity and are fired with coal. The lead bullion comes in on a high line in carload lots of 30 to 40 tons. The entire lot is trucked from car, weighed over scales and thrown into the sampling kettle, which has been previously cleaned. The kettle is fired and the bullion melted, then heated up until no frozen bullion remains in the bottom or adheres to the sides of the kettle. The kettle is scraped clean, sides and bottom, and the wet dross is skimmed off clean into molds. The kettle is stirred thoroughly during the period in which 24 to 30 samples of approximately ½ assay ton are taken. These samples are taken in a long-handled iron mold, providing for taking six to eight "gumdrops." The mold is inserted and heated to the same temperature as the molten metal, and gumdrops are dipped out and cooled in the mold by dipping the bottom of the mold into water. The samples should be free from fins. The gumdrops represent the contents of the kettle, and in assaying, they should not be clipped, but weighed up and the result computed. The dross bars are weighed and sawed according to old hammer and punch sample template. The sawdust represents a sample of the dross. In cases of high gold in the bullion there is a slight correction in the assay.

This method of sampling was developed at the Omaha plant of the American Smelting & Refining Co., W. T. Page, Manager, in 1903 or 1904, for adjusting differences in checking smelter sampling.

L. S. AUSTIN.—The base bullion which you treat is very clean. It has been remelted, and skimmed before shipment. How rapidly can you get it through the softening furnace, and how does it compare with ordinary base bullion?

G. P. HULST.—Twenty-four hours is the softening limit on these charges. In other words, it is charged into the furnace, heated up, and the first and second skimmings are taken off, and it goes into the kettle the next day; that is, 24 hr. elapse from the time the bullion is charged to the time it is in the kettle.

L. S. AUSTIN.—It seems to be a long time to hold such a pure bullion. I suppose, though, it comes around conveniently so that you have a time for all the other operations of refining. Isn't that the case?

MR. HULST.—Every refinery is based on those principles.

MR. AUSTIN.—Well, in other words, you do not tap your softening furnace as soon as it is softened.

MR. HULST.—Absolutely not.

MR. AUSTIN.—Of course it is convenient for all operations to come around at the same time of the day. Now, another question: You say that you ship away the matte you make to the smelter. To what smelter?

MR. HULST.—To the Perth Amboy smelter of the American Smelting & Refining Co.

Electrical Fume Precipitation at Garfield

BY W. H. HOWARD, GARFIELD, UTAH

(Salt Lake Meeting, August, 1914)

As the result of a series of analyses and volume determinations of gases discharged from the converters at the Garfield Smelting Co.'s smelter at Garfield, Utah, it was found that a considerable amount of lead fume was escaping from the stacks of the converter plant, and that if this lead could be recovered at a reasonable expense an attempt should be made to collect it.

Two methods suggested themselves: (1) filtration through bags; (2) electrical precipitation of the condensed fume by a Cottrell plant.

Filtration through Bags

Tests were made in filtering converter gases through the ordinary woolen bags of lead-smelter bag houses to determine how long such material could withstand the corrosive action of the gases. While blowing on white metal only, the fabric was usually destroyed in 10 to 14 hr. In tests conducted in filtering gases when blowing on matte or slagging stage exclusively and when relatively considerable lead was being volatilized, bags in one instance lasted 60.5 hr. It therefore was evident that the Garfield converter gases did not contain in themselves sufficient neutralizing elements for the purpose of direct bag filtration, even during the slagging stage.

Such results were more or less anticipated, but it was thus proved that under all conditions of blowing Garfield mattes a very considerable amount of neutralizing agent, zinc oxide or lime, or both combined, as in the Sprague process, would be required to render bag filtration possible. Again, owing to the constantly changing quantities of SO_3 from a given battery of converters, depending on the number blowing at a given time and on the stages of the blow, it would be practically impossible to feed automatically the required amount of neutralizer. In order to safeguard the bags sufficient neutralizer would have to be continuously added to meet the maximum SO_3 output, thus wasting neutralizers and giving an undesirably diluted final product. Such a condition is entirely different

from what would prevail in a battery of roasters, where a fairly constant volume of gas of fairly constant tenor can usually be maintained.

Having calculated the amount of neutralizer required, it was estimated that the cost of a bag house and its operation and the treatment of the resultant diluted product would not prove a commercially satisfactory proposition.

The Cottrell Process

As the complete elimination of SO_3 under existing conditions at Garfield was not imperative, a method which would recover the valuable constituents of the gases was the one to be considered.

The volume of gas, including entrained air at the converter hood produced from each Peirce and Smith converter, calculated at standard temperature and pressure, was found to be approximately 25,000 cu. ft. per minute, or an average of over double that volume at flue temperature.

An average of four to five converters is operated under normal plant conditions and practically never are over four blowing simultaneously, equivalent to a production of 200,000 cu. ft. of gas per minute at flue temperature for the whole department.

The ratio of SO_3 to SO_2 has been very variable, depending on the character of the matte treated and on the period of the blow. The average of several determinations showed 1 to 18, but at times it has been as high as 1 to 8.

The Cottrell process described in its simplest terms consists in the conversion of a high-tension alternating current of 30,000 to 60,000 volts into a direct intermittent current of the same potential, and the application of this current to certain electrodes suspended in a flue conveying the gases to be treated. These electrodes in their simplest form consist of narrow suspended metal plates regularly spaced 5 to 12 in. apart, depending on voltage, and between adjacent plates, parallel and equidistant from them, is stretched a fine wire to every space, the wires forming the discharge electrodes and the plates the collecting electrodes. There is thus maintained through a silent or glow discharge an electrically charged field through which the gases travel and all dust or condensed fume or moisture becoming electrified is deposited on the plates.

Cottrell Plant and Work at Coram, Cal.

From an inspection of the Cottrell plant then installed at the works of the Balaklala Cons. Copper Co., at Coram, Cal., it was thought that the Cottrell process might be advantageously used for the recovery of the lead fume from the converters at Garfield; if not completely, then to an extent that might prove profitable. The treaters consisted essentially of chambers in which were suspended sheet-iron plates 6 in. wide by 10 ft. long and

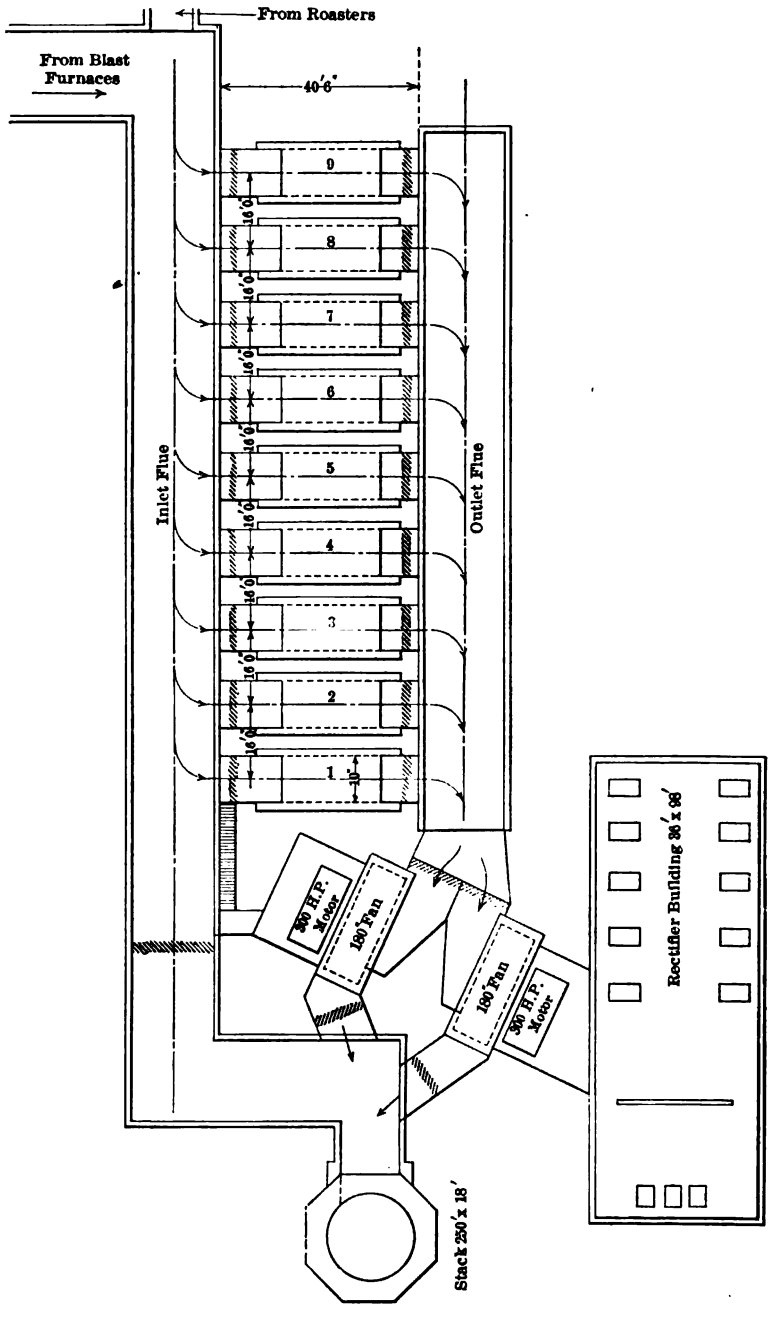


FIG. 1.—GENERAL ARRANGEMENT OF THE COTTRELL PRECIPITATING PLANT OF THE BALAKLALA CONS. COPPER CO., CORAM CAL.

5 in. apart, these forming the collecting electrodes. The discharge electrodes consisted of two iron wire strands in which were twisted pieces of asbestos fiber, forming numerous points for the discharge of a high-ten-

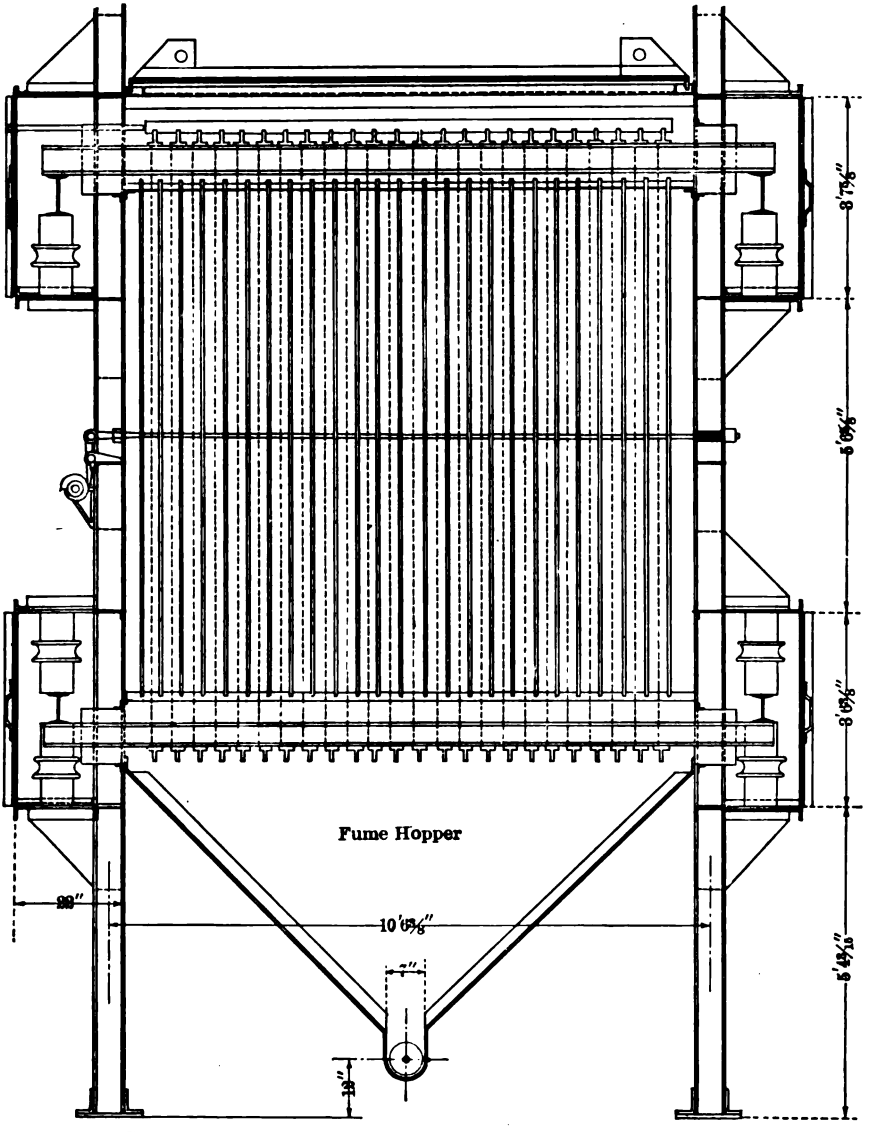


FIG. 2.—SECTION THROUGH COTTRELL PRECIPITATING UNIT, BALAKLALA CONS. COPPER CO.

sion direct intermittent current of 28,000 to 30,000 volts. By referring to Figs. 1 and 2 it will be seen that the installation consisted of nine units.

eight operating and one being cleaned, and it might be supposed that a proper distribution could not be secured with such an arrangement as shown, but with louver dampers at the ends of each unit a fairly even flow through all units was obtained. The chief mechanical defects in a unit of this description are the spaces between the ends of the plates and the roof, and also along the bottom of the unit, where gases would escape treatment.

I was informed that the apparatus worked best with gases between 120° and 140° C. A temperature approaching but not less than 120° seemed the most desirable, though evidently, in the light of our present knowledge, not sufficiently low to condense all smoke-forming elements, which on being discharged into the atmosphere would become visible as smoke.

On my visit there in the spring of 1911 the recovery of solids from the smoke was usually 80 to 90 per cent., an attainment not sufficient to meet the demands of the courts. It was my belief, however, that if an extension of time had been allowed the elimination required would have finally been accomplished.

Preliminary Experimental Work at Garfield

Actuated by this faith in the possibilities of the process to successfully treat converter gases, it was decided to carry on experiments at Garfield with a view to determining the best working conditions as to temperature, acidity, etc., and to discover the most efficient form of electrodes for the purpose. This work was intrusted to Mr. Rathbun, formerly chief electrician at the Balaklala smelter, who was fully familiar with the process as conducted there, and R. F. Barker, field chemist at Garfield.

This experimental work was started in August, 1911. Equipped with the necessary apparatus to produce a 20,000 to 30,000 volt direct intermittent current, the regulation plates and pubescent electrodes were first tried. Then iron pipes of different diameters and of varying lengths were experimented with. It was assumed that a pipe with an axially placed wire would give more uniform electrical treatment than the plates and wires, and the experiments proved this. With the current available of 30,000 volts, it was finally determined that 5-in. pipes 10 ft. long placed vertically gave the best clearances if the velocity of the gases treated did not exceed 15 ft. per second through the pipes; and finally the pubescent electrode, which was a source of considerable trouble, was replaced by a No. 22 gauge steel wire.

Pipes of larger diameter with the application of higher voltage were considered, but it was thought that the smaller pipe would secure better distribution; and having proved the efficiency of the 5-in. pipe it was

decided to adopt that size, leaving the use of large-pipe and high-voltage installations to future development.

50,000 *Cu. Ft. Converter-Gas Treater at Garfield*

Having proved the greater efficiency of the pipe as compared with the plate as a collecting electrode and substituting a single wire to replace the regular pubescent discharge electrode, a plant was designed for the treatment of 50,000 cu. ft. of converter gas per minute.

The treater, as illustrated in Figs. 3 and 4, consisted of 608 5-in. iron, pipes (well casing), 10 ft. long, placed vertically over corresponding openings in a floor, similarly to the flues of a vertical boiler. Centrally, through each pipe, was stretched a No. 22 gauge steel wire. The treater was divided electrically into four sections so that different voltages could be applied to different parts of the plant.

Current was taken from a 2,300-volt alternating-current line stepped up to 23,000 to 30,000 volts. This high-voltage alternating current was by means of revolving rectifiers converted into an intermittent direct current of the same potential for the precipitating electrodes. The rectifiers were run by three-phase synchronous 220-volt motors, power to operate which was taken from the same source as that converted at the rectifiers, but stepped down to required voltage.

On the assumption of the treatment of 50,000 cu. ft. of gas per minute through 600 pipes, the average velocity through the pipes would be about 10 ft. per second, or the gases would be under treatment for 1 sec. in the 10-ft. pipe.

On Dec. 6, 1912, the gases from one converter were turned into the treater, being drawn through it by the draft of an adjacent stack 150 ft. high. It was discovered in the preliminary experiments that during blowing on white metal to copper, when a sufficient amount of sulphuric acid was present in the gas, the coating on the pipes from the condensed fume was conductive and the process offered no difficulties; but that during the slagging period, with a small amount of SO_3 present, a non-conductive coating was formed in the pipes and after a short time the current was interrupted and further precipitation ceased. It was found, however, that by the injection of a small amount of moisture in the gases before reaching the treater the coating in the pipes was rendered conductive and the process worked satisfactorily at about 90° C. With mixed gases from the slagging and white metal stages, as would occur under normal operations with three or more converters running, no addition of moisture was found necessary. But being limited to the treatment of the gases from one converter, fine sprays of water were injected into passing gases during the slagging stage, amounting to about 3 to 5 per cent. of moisture in the dust collected. Owing to occasional arcing

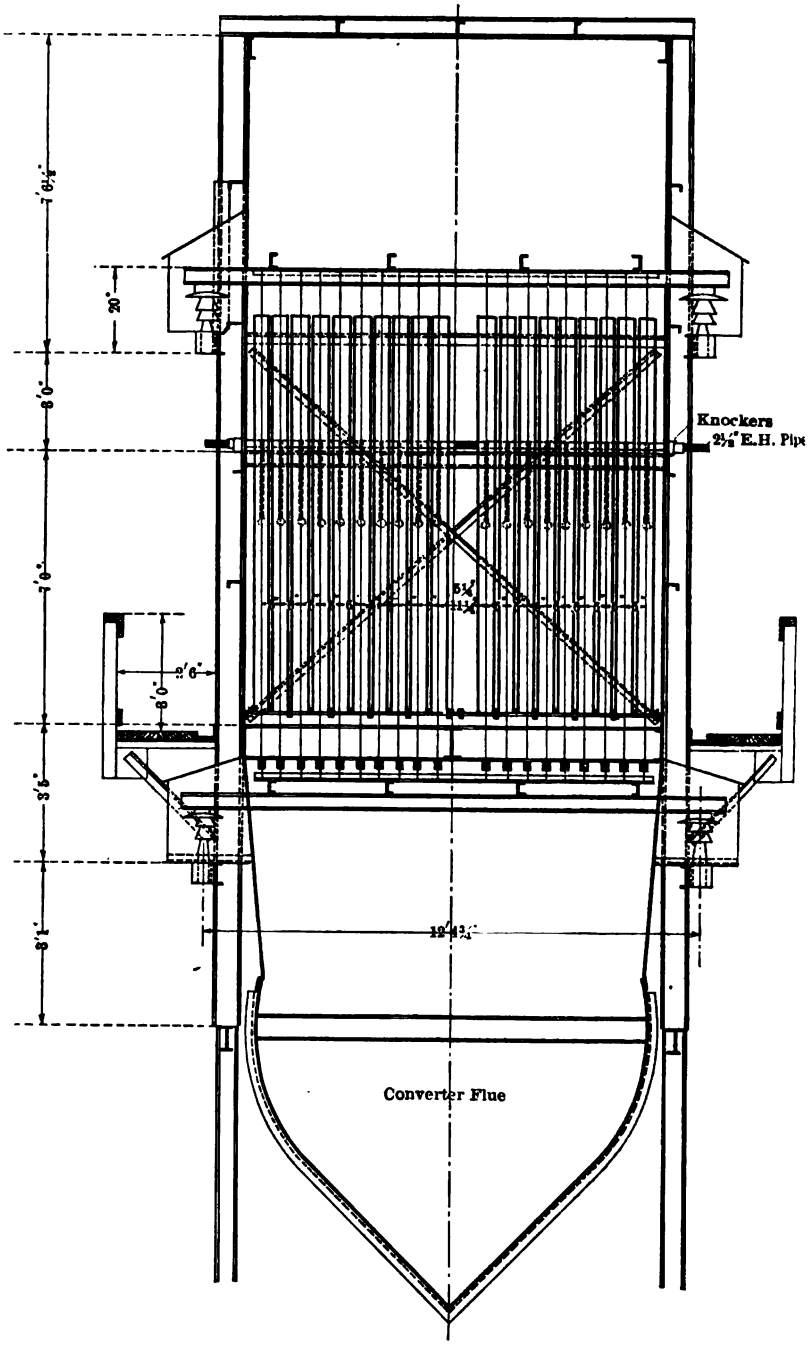


FIG. 3.—TYPICAL CROSS-SECTION OF EXPERIMENTAL COTTRELL PRECIPITATING PLANT OVER CONVERTER FLUE.

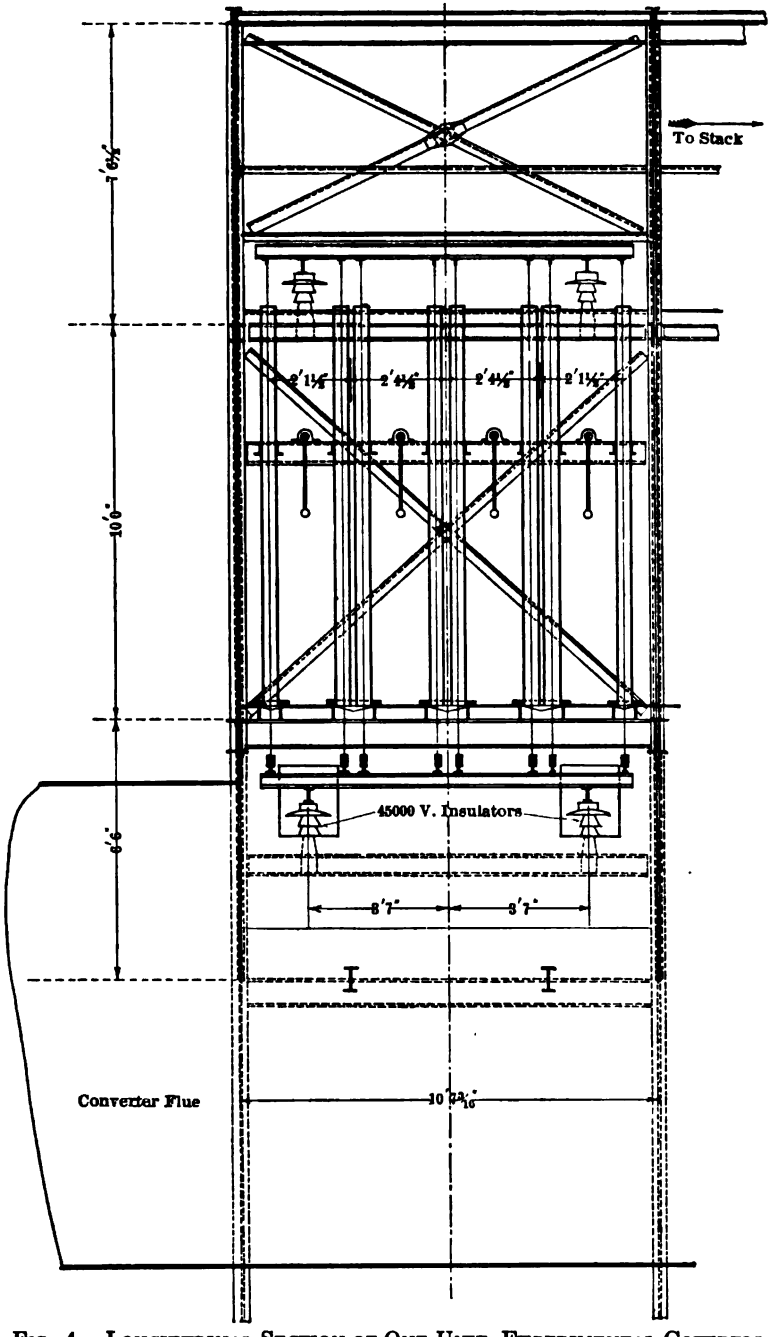


FIG. 4.—LONGITUDINAL SECTION OF ONE UNIT, EXPERIMENTAL COTTRELL PRECIPITATING PLANT.

the No. 22 wires were at times burned off, but with No. 14 gauge wire this trouble was practically eliminated.

The plant was run continuously for several weeks and from analyses of the escaping gases it was determined that the average recovery of lead was 97.25 per cent. The dust collected ran 41 per cent. lead, mostly in the form of basic sulphate.

During the progress of the work it was observed that during the blow on white metal, where a conductive deposit was obtained in the pipes, practically all of the lead was precipitated from the smoke at a temperature of 340° C., but that the SO_3 and As_2O_3 escaped. To precipitate the latter and obtain complete smoke clearances a temperature of about 90° C. was necessary. This suggested the possibility of fractional precipitation by using treaters in series, whereby passing gases could be treated at different temperatures, thus classifying the elements of the smoke. This gave an aspect to the possibilities of the process not hitherto considered.

As the result of the recoveries made in the 50,000 cu. ft. installation it was decided to apply to the Executive Committee of the company for authority to erect a plant to treat all the smoke from the converters, this plant to be an adjunct to a new flue system then under consideration, which will later be referred to.

Pending the granting of the required authority it was decided to erect a small treater for experimental work on blast-furnace, MacDougall-roaster, and reverberatory gases.

Experimental Plant for Treating Blast-Furnace, MacDougall, and Reverberatory Gases

The treater, which consisted of 152 5-in. pipes and the necessary electrical equipment, was erected about 300 ft. from the main smelter stack, or approximately 2,700 ft. from the blast furnaces, 1,800 ft. from the MacDougall-roaster installation, and 1,700 ft. from the reverberatory furnaces, where the three flues of these departments run close together and parallel on to the chimney. A 30-in. sheet-iron pipe with the necessary branches and dampers was connected with a fan, whereby the required gases for test could be withdrawn from any flue and delivered to the treater.

The tests conducted on these gases were with the view of not only recovering values, but also eliminating all smoke-forming substances, such as sulphuric acid and moisture. Under such requirements of perfect or high percentages of clearances, lower temperatures and lower velocities than shown for converter-gas treatment for lead values became necessary.

In the tests (Tables I and II) gases were continuously sampled during process of treatment. The apparatus was run in the day time only, in

Table I.—Tests on Blast-Furnace Gases. Three Furnaces in Operation. Mar. 28 to April 3, 1913

Test No.	1	2	3	4	5	6	7
Length of test, minutes	500	530	430	540	535	450	500
Temperature of gas entering treater, degrees C.	95	84	85	84	115	84	90
Average temperature of gas in electrodes, degrees C.	85	70	70	70	90	70	75
Velocity of gas through electrodes, feet per second	5	3	3.3	3.4	7.2	7.1	7.3
Total volume treated, cubic feet	3,100,000	1,966,300	1,780,200	2,257,200	4,761,500	3,960,000	4,550,000
a Clearance, per cent.	98	Perfect	Perfect	Nearly perfect	92	95	93
Total volume of B. F. gas passing to stack, cubic feet per minute	280,000	290,000	290,000	292,000	282,000	291,000	284,000
Per cent. of this volume treated	2.48	1.54	1.74	1.73	3.68	3.5	3.65

a Test No. 1. Clearance, 98 per cent. The solids escaping were mostly moisture and some sulphuric acid.

Tests Nos. 2, 3. Gas was perfectly clear. All dust, sulphuric acid, and moisture collected.

Test No. 4. All dust collected, but very slight amount of sulphuric acid and moisture escaped.

Test No. 5. Clearance, 92 per cent. At the higher temperature and velocity in this test practically all dust was collected, but some acid and moisture escaped.

Tests Nos. 6, 7. Practically all dust collected, but not all acid and moisture on account of higher velocities. Clearance 95 and 93 per cent.

Table II.—*Test for Nine Days on MacDougall-Roaster Gases*

Test No.	1	2	3	4	5	6	7	8	9
Date, April, 1913	11–12	16–17–18	21	22	23	24	25	26	27
Length of run, minutes....	920	1,397	500	520	530	520	545	530	300
Temperature of gas entering treater, degrees C....	110	85	85	50	70	70	94	100	84
Average temperature of gas in electrodes, degrees C.	90	70	70	35	55	55	80	85	70
Velocity of gas through electrodes, feet per sec.	7.6	7.5	5.3	3.0	3.2	3.4	8.3	8.5	8.2
Total volume treated, cubic feet.....	8,638,800	12,992,100	3,275,000	1,934,400	2,098,800	2,173,600	5,613,500	5,565,000	3,039,000
a Clearance, per cent.....	96	Perfect	Nearly perfect	Nearly perfect	Perfect	Perfect	97	Nearly perfect	Nearly perfect
Total volume roaster gas passing to stack, cubic feet per minute.....	327,000	342,000	349,000	371,000	343,000	342,000	346,000	323,000	315,000
Per cent. of this volume treated.....	3.19	3.08	2.21	1.32	1.37	1.45	3.34	3.56	3.66

a Tests Nos. 1 and 7. All dust precipitated, but not all H_2SO_4 and moisture.

order to have Mr. Barker's full attention and thus enable him personally to see that both operating and analytical work would be carried on in a manner leaving no doubt as to the accuracy of the results.

From an inspection of the tables it will be seen that to obtain perfect clearance a velocity of about 3.5 ft. per second in the pipes was necessary. In special experiments with gases diluted with air perfect clearances were obtained with 7 ft. per second in the electrodes. This is equivalent to reducing the dust particles per unit of volume, and such a condition would be obtained by passing gases through a proper settling chamber preparatory to Cottrell treatment.

Several days were devoted to the treatment of mixed gases from the reverberatories, roasters, and blast furnaces, with results equally as good as shown for blast furnaces and roasters separately.

The apparatus worked generally well electrically under most conditions. Different kinds of deposits were obtained, wet and dry, and in some cases wet mud dripped from the pipes, but very little trouble was experienced in making the apparatus work.

Summary of Tests at Garfield

The results of the experiments of December, 1912, to April, 1913, were:

1. The establishment of the pipe electrode, now being generally adopted in recent Cottrell installations.
2. The replacement of the pubescent electrode by a single wire.
3. Proof that 95 per cent. to practically complete clearance of Garfield smelter gases can be accomplished if temperatures are maintained under 100° C.
4. That by treatment of gases at different temperatures, by treaters in series with the necessary drop in temperatures between treaters, fractional precipitation may be possible. For example, collecting the bulk of the lead in one treater and then by the further cooling of gases collecting most of the arsenic, sulphuric acid, etc., in a second treater.

Experiments at Murray Plant, American Smelting & Refining Co.

The method of fractional precipitation has since been tried experimentally on the treatment of roaster gases at the Murray plant of the American Smelting & Refining Co. Dust, acid, and moisture carried by the gases produced a sticky, muddy deposit on the electrodes when tried in one treater, and the difficulty of removing this mud became a problem. Two small treaters were then placed in series. The temperature of the gases entering the first treater averaged 125° C. and entering the second treater 70° C. In the first as high as 95 per cent. of the total

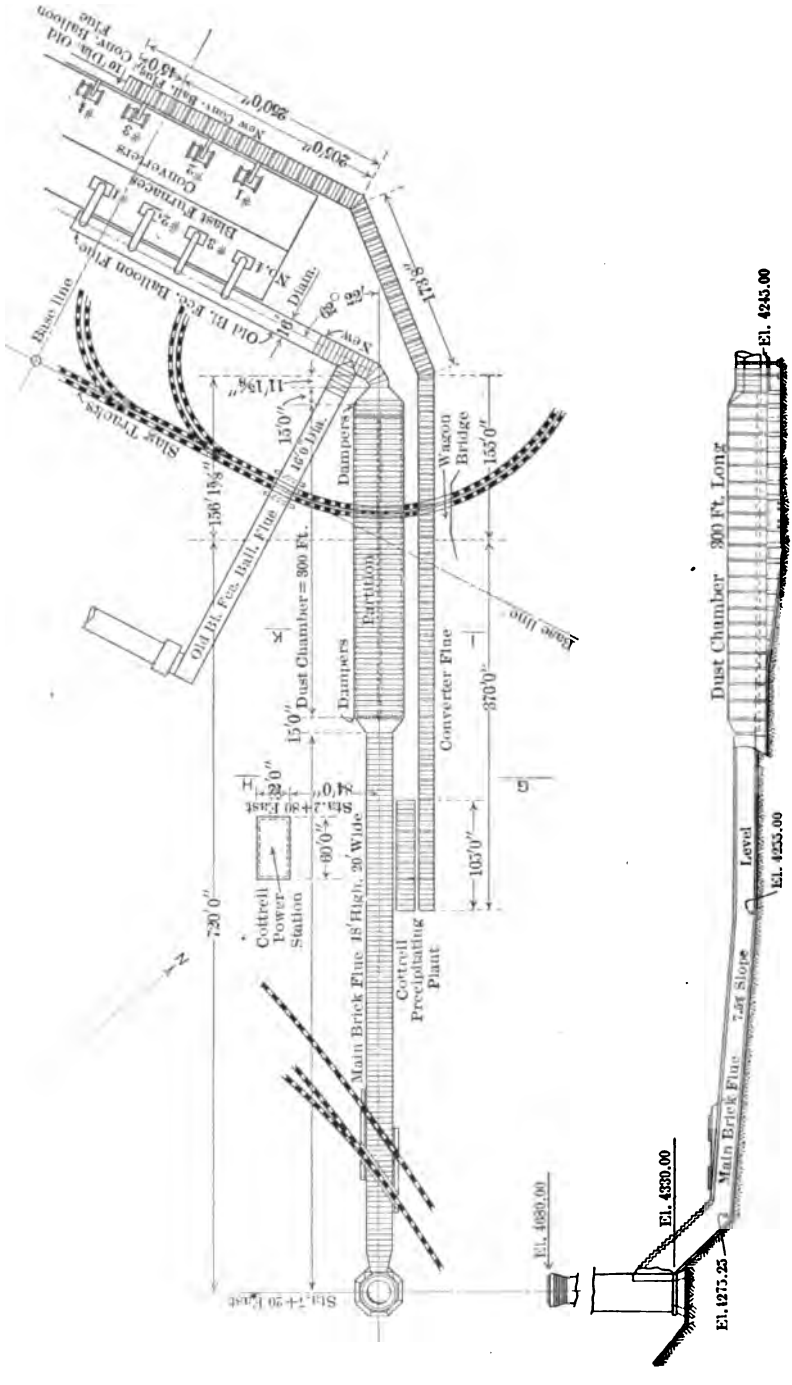


FIG. 5.—CONVERTER AND BLAST-FURNACE FLUE SYSTEM, GARFIELD SMELTER.

precipitate was collected, in a comparatively dry condition. In the second treater most of the remaining dust, acid, and water, approximating 4 per cent., was collected, sometimes sufficiently moist to drip from the pipes. These results have been encouraging and further experiments will be tried. It is too early to state that this problem has yet been completely solved, as the maintenance of proper temperatures is of critical importance to the work.

I wish to emphasize that the results obtained at Garfield and at Murray were on the treatment of gases particularly favorable to the process and it does not follow that all smelter smoke can be treated as successfully. Again, while experimental results have warranted our trying the process on a large scale, we have no definite assurance that the element of time may not develop difficulties not now apparent.

Cottrell Plant for Treatment of Gases from Converter Plant

Fig. 5 is a general plan of a new flue system for the blast-furnace and converter gases, built primarily on account of the inadequacy of

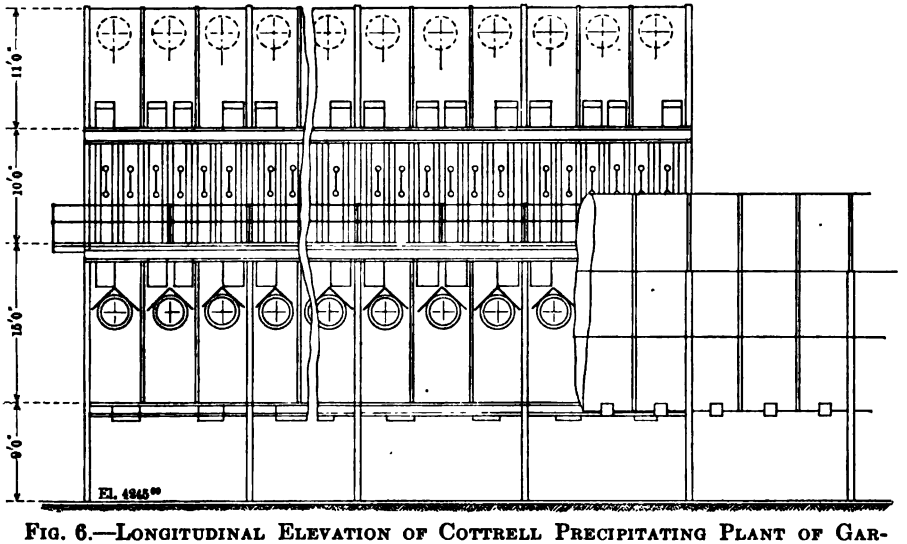


FIG. 6.—LONGITUDINAL ELEVATION OF COTTRELL PRECIPITATING PLANT OF GARFIELD SMELTING CO.

the old flue system to handle properly all the gases of the plant under the present output.

For the collection of dust from the blast furnaces the gases will pass through a dust chamber 300 ft. long by about 920 sq. ft. cross-sectional area, about 100 ft. of which will be filled with suspended wires as followed in the very large chamber of the Boston & Montana Reduction Works at Great Falls with such excellent results.

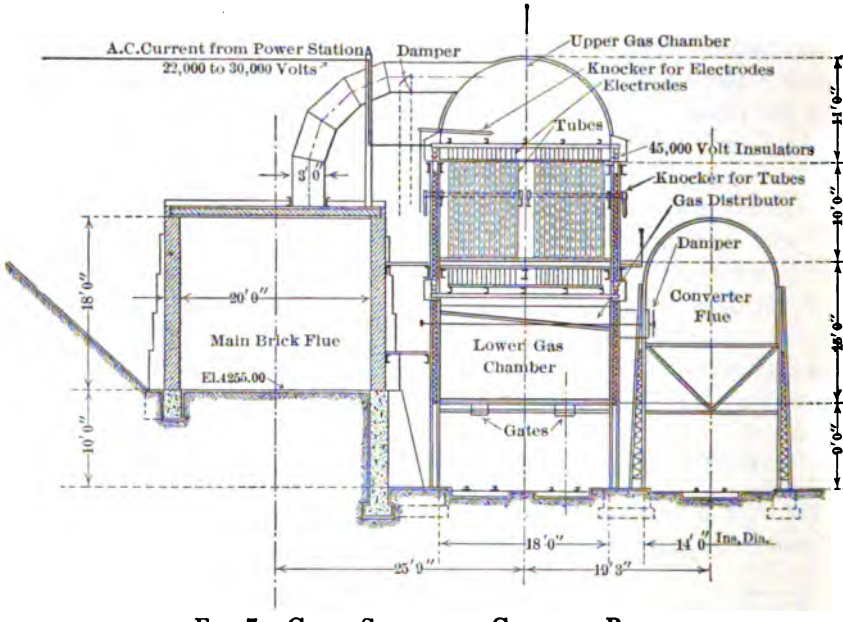


FIG. 7.—CROSS SECTION OF COTTRELL PLANT.

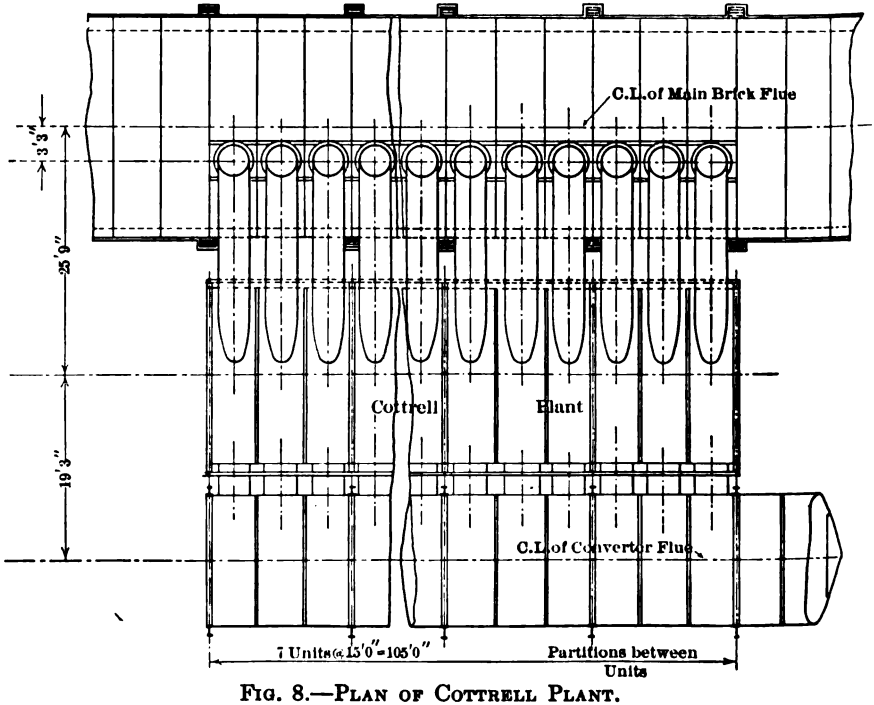


FIG. 8.—PLAN OF COTTRELL PLANT.

From the chamber the gases pass up a flue connecting with a chimney 350 ft. high by 22 ft. internal diameter at the top, built on a solid rock foundation 85 ft. above the general smelter level, or equivalent to a total height of 435 ft.

The converter gases which are destined for Cottrell treatment for recovery of lead fumes, etc., will pass through a long steel flue of 210 sq.

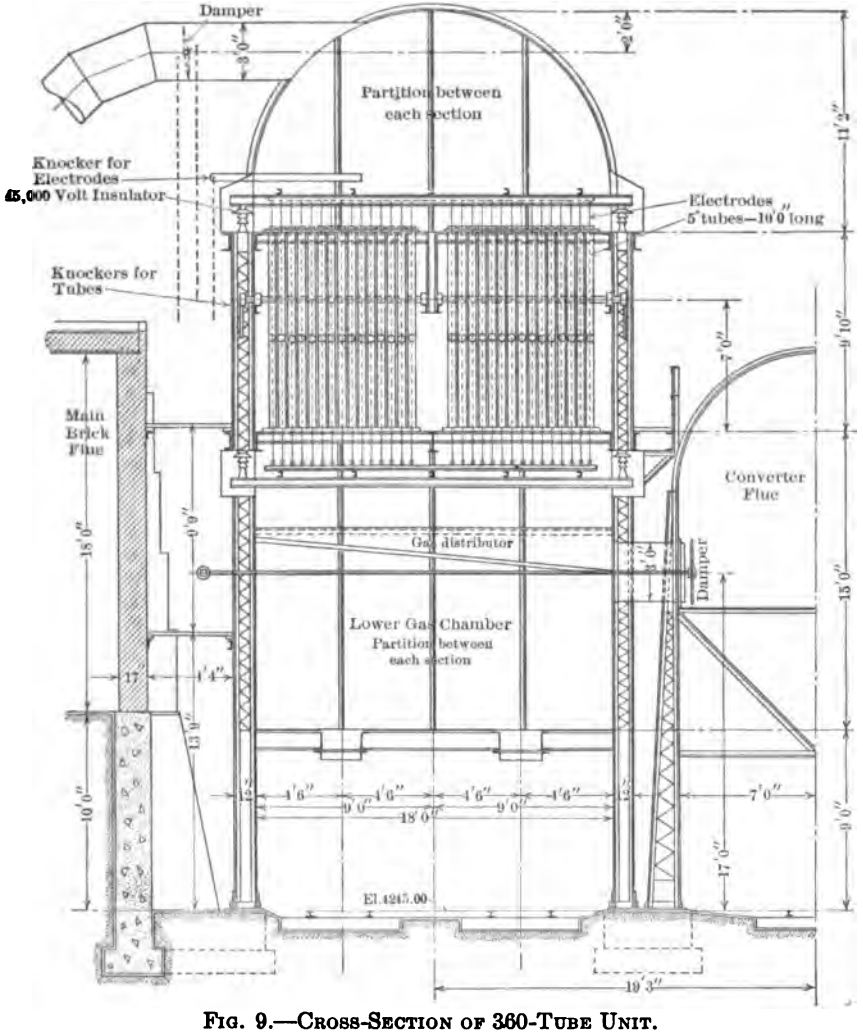


FIG. 9.—CROSS-SECTION OF 360-TUBE UNIT.

ft. cross-sectional area and about 1,000 ft. long, which will collect practically all cuprous material from the smoke, and the gases in their passage will become sufficiently cooled for delivery to the electrical precipitators, as illustrated in Figs. 6 to 10.

Electrical Precipitator.—The electrical precipitator consists of seven units, containing 360 5-in. pipes, 10 ft. long, per unit, or a total of 2,520 for the whole installation. Six units will be usually operated while one is being cleaned.

On the basis of seven operating, with an output of 250,000 cu. ft. of gas a velocity of 12 ft. per second will be obtained; on the basis of

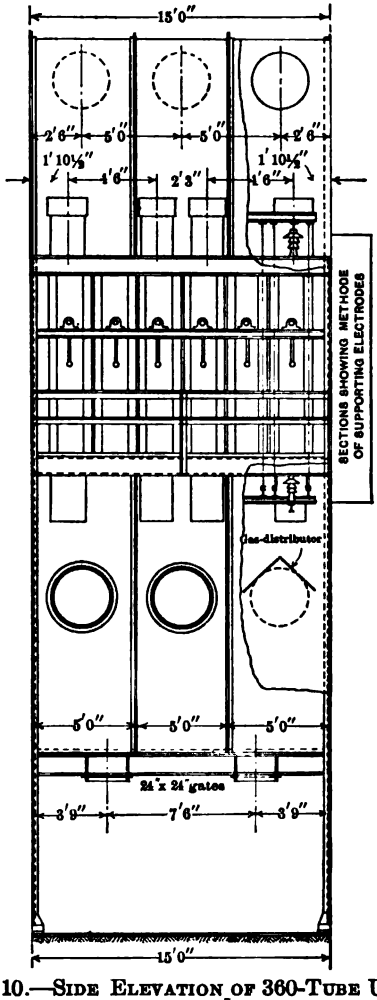


FIG. 10.—SIDE ELEVATION OF 360-TUBE UNIT.

200,000 cu. ft. a velocity of 9.6 ft. per second, and with six units on the same volume 11.2 ft. per second, all within the required velocity for the special treatment of converter smoke. Certain changes contemplated in converter hoods will further reduce the volume of gas delivered to the treater.

Gases from the converter flue will be delivered to the lower chambers of treater through 36-in. pipes, three pipes per unit, each pipe having a deflector extending across to insure the best possible distribution. After passing through the pipes the gases reaching the upper chamber then pass through 36-in. pipes to the main flue leading to the chimney, there being three pipes per unit, and again tending to the maintenance of an even flow through the electrodes.

Transformer and Rectifier House.—With the apparatus used in the single converter gas plant of 608 pipes referred to it was estimated that the maximum power used was 25 h.p. In determining the size of transformers necessary for the new installation this figure was used as a basis. As the units consist of 360 pipes each and as it may be necessary at times to connect two on the same transformer, it was decided that 20-kw. transformers would have liberal capacity for any condition that might arise. The power necessary for the treatment of 250,000 cu. ft. of gas per minute will probably amount to 100 h.p.

Electrical Apparatus.—This includes seven complete motor-generator-rectifier sets and transformers, and switchboard panel to control these.

A set consists of a 30-h.p., 250-volt, direct-current motor, direct connected to a 20-kw., 4-pole, 60-cycle, 220-volt single-phase alternating-current generator. The 220-volt current is stepped up to 20,000 to 30,000 volts by means of a 20-kw. transformer and then led back to the rectifier, which is on the same shaft with the motor-generator. Here it is rectified before going to the treater. The voltage before going to the transformer may be regulated to any desired value by varying the field excitation with a rheostat located at the switchboard.

On the switchboard are located the necessary switches and instruments for the motor-generator sets. Each generator circuit is supplied with an automatic circuit-breaker switch which can be set to carry a short circuit for a definite length of time before opening.

It is expected that the plant will be completed some time in July, and I believe it should accomplish what it has been designed for; that is, the collection of those elements in the smoke which may be of commercial value.

Discussion

Edgar M. Dunn, Anaconda, Mont.—Mr. Howard's contribution to the literature on the Cottrell method of electrostatic precipitation of flue dust and fume at Garfield has been read by us with peculiar interest for the reason that we have been experimenting with the same process at Anaconda for the past eight or nine months, with a view to adapting it, under the direction of Dr. Cottrell and the supervision of J. O. Elton and D. R. Kellogg, representing the Anaconda Smelter Commission, to the treatment of very large volumes of gases, using large pipe treaters and

various forms of electrodes, with about 150,000 volts rectified current. For the amount of gas to be treated here the 5-in. pipe means a cost for installation that is practically prohibitive, due to the tremendous number of pipes required, and we have been trying to cut installation costs by increasing the size of the pipes. All of our work is still in the experimental stage, but we have obtained good clearance on our mixed roaster and converter gases with a 3-ft. pipe treater 20 ft. long, and with 5 ft. velocity therein; and with a discharge electrode of a single No. 29 nichrome wire set in the middle of the 3-ft. pipe, with a consequent electrode spacing of 18 in. The current used has been 150,000 to 160,000 volts, and of course there have been troubles galore, relating particularly to insulation and rectifying. Blast-furnace gases did not give so good clearance, probably due to relative poverty in moisture and SO_3 content of blast gases to roaster-converter gases; making the blast gases the poorer in conductivity. Our experience has been so satisfactory that in our next experimental plant—built on a somewhat larger scale—we shall attempt a 4-ft. treater with greater electrode spacing, and a somewhat higher voltage—up to 220,000.

As to fractional precipitation, mentioned by Mr. Howard, this idea was tried out at Anaconda last year (1913).

Formerly our furnaces (Bruntons) roasting flue dust for the recovery of white arsenic, passed so great an amount of dust to the settling kitchens that refining of this arsenical product was necessary to eliminate the 5 to 20 per cent. dust contained therein. Consequently the manufacture of As_2O_3 was effected in two stages, with a reverberatory roast of the first product to obtain the final 99.5 per cent. As_2O_3.

Part of the dusting trouble was due to too high a temperature and draft in the first roasting furnace, and we are now installing new furnaces of another type to volatilize the arsenic trioxide in the dust recovered from our flues. This process is simply one of volatilization (physical change) of the As_2O_3 present in the flue dust; no chemical change of an appreciable nature occurs in the best practice, the arsenates contained in the dust not being commercially reducible below a smelting temperature.

Fractional precipitation was used last December to separate the As_2O_3 from the dust carried along from the first roasting furnace in the gas stream. Two treaters were used. In the first—gases entering hot, 310° C.—practically all of the dust was precipitated; while in the second —gases cooled to 90° C. by admission of air—white arsenic of 99.7 per cent. was directly thrown down; fractional precipitation thus eliminating the refining furnace formerly necessary.

Our new flue-dust roasting furnaces will be equipped with fractional precipitation treaters, thus saving the cost of refining the crude product; their operation must still be regarded as experimental, due to change in

type of roasting furnace. We hope to furnish complete results to the Institute at an early future date.

Finally, we note with regard to Mr. Howard's report of experiments at Murray on fractional precipitation, that "as high as 95 per cent. of the total precipitate was collected" in the first treater at an average entering temperature of 125° C.; while in the second treater (average entering temperature 70° C.) "most of the remaining dust, acid, and water, approximating 4 per cent., was collected."

This might lead one to believe that the first treater was but 95 per cent. efficient. A moment's thought, however, will bring out the fact that the efficiency of the first treater, *at its temperature*, must have been close to 100 per cent.; the change in vapor tension between 125° and 70° rendering additional acid and water precipitable at the lower temperature of the second treater. In other words, had the gas contained vapors capable of precipitation between 70° and atmospheric temperature, a third treater (or a lowering of the temperature in the second treater) would have recovered those vapors in the liquid or solid state.

F. G. COTTRELL, San Francisco, Cal.—There is very little I can add to Mr. Howard's paper except a word of appreciation for his energy and ability in carrying this work through to completion. As most of you doubtless remember, the work on this same general subject which was under way several years ago at the Balaklala smelter, in California, was terminated by that smelter shutting down on account of fume troubles, in which at the time no very definite segregation was attempted as to the part played respectively by dust and sulphur dioxide.

Although that installation ran for only a few months, and was really never thoroughly completed, it served to bring the process forward to a scale large enough to give an idea of what could be done in the collection of values, and Mr. Howard was the man among the members of the smelting profession who took enough interest in the work, saw the possibilities with sufficient clearness, and had the courage, in the face of the skepticism of many of his associates and official superiors, to take the work up purely as a matter of saving values. The process thus entered on this second stage which he has so admirably carried through at Garfield.

Looking backward on the history of the work, it appears doubtful whether it would ever have lived through the difficulties encountered in its early stages had it not appeared as a possible solution of the life and death struggle some of the plants were then making in the courts on fume-damage suits. But once these early difficulties were passed and the work was placed on a reasonably definite and certain basis as an established method, its general importance to the industry and the interest now taken in it have rapidly come to center chiefly around the collection of values previously lost in the smoke.

The work at Garfield, as you know, is aimed primarily at the saving of values from the converter flue. The degree of recovery attained at the Balaklala plant appealed to Mr. Howard as amply warranting, even at that stage, its operation on converter flues, aside from any question of fume nuisance. In the present plant, with the lack of any direct disturbance from that cause, it should be possible to gain a thorough knowledge of what can be done on the recovery of values. In the meantime, of course, as Mr. Dunn has told you, the work has gone on at other places, largely due, I feel sure, to the stimulating effect of Mr. Howard's own confidence and example. A plant nearly the size of that at Garfield is already running at the Trail lead smelter in British Columbia.

I feel that Mr. Howard is to be very heartily congratulated on the successful way in which he has carried out this work in the face of a great many difficulties, some of which I have had opportunity to see and know as the work progressed.

I have with me a few samples of the material collected on the Anaconda work which I thought might be of interest in this general connection, as they illustrate in a rather striking manner the possibilities of fractional precipitation suggested by Mr. Howard in his paper and further emphasized by Mr. Dunn's discussion. In this set [exhibiting samples] there are three bottles numbered 1, 2, and 3. No. 1 is the original feed to the arsenic furnaces in one of the experiments of Anaconda; No. 2 is the dust precipitated in the first precipitating unit—that is, precipitating the dirt out of the arsenic at a temperature still retaining the greater part of the arsenic in the form of a gas; and No. 3 is the refined arsenic precipitated in a subsequent electric treater after chilling by admixture of cold air. The gray of the original flue dust, the almost jet black and the snow white colors respectively of the impurities removed and the refined arsenic obtained serve to make this, as you notice, a very good illustration of the principle.

The Bag House in Lead Smelting

BY H. H. ALEXANDER, MAURER, N. J.

(Salt Lake Meeting, August, 1914)

In the early part of the last century textile fabric was used for the filtration of products of combustion and lampblack was obtained by passing smoke through a series of canvas bags. Natural draft was used to draw and force the smoke through the bags. About 1850 bag filtration was used for collecting zinc oxide. Around 1876 it was introduced for collecting the fume from lead-ore hearth smelting in Missouri. Shortly afterward it was used, first at Portland, Me., and later at Cañon City, Colo., for the collection and production of zinc-lead pigment in treating mixed zinc-lead sulphides. In 1890 the Globe Smelting & Refining Co., at Denver, Colo., installed a bag house containing 1,458 bags for the recovery of fume from silver-lead blast furnaces. This installation was too small to handle all of the gases and was increased to approximately 2,300 bags. In 1900 the number of bags was again increased, making a total of approximately 2,800.

The usual difficulties incident to new installations were encountered, but they were gradually overcome and the bag house became accepted as standard practice for this class of smelting. As all of the bag houses built since have followed the general lines of the Globe installation, a short description may be of interest.

The general construction of the Globe bag house is shown in Figs. 1, 2, and 3.

The building is 149 ft. 8 in. long by 67 ft. 4 in. wide and 40 ft. 2 in. high from the basement floor to the eaves of the roof, having brick walls 26 in. thick from the foundation up to the thimble floor, from there an 18-in. wall for 16 ft. and a 13-in. wall from this point to the top, a distance of 14 ft. 2 in. Additional stiffening is given to the walls above the basement by pilasters located every 16 ft. The end-wall construction is similar to that of the side walls with the exception of the spacing of the pilasters, which is somewhat greater. The roof is made of corrugated iron laid on 1-in. boards and supported by a timber frame work spaced 16 ft. apart resting on the basement partition walls. On top is a louver running the

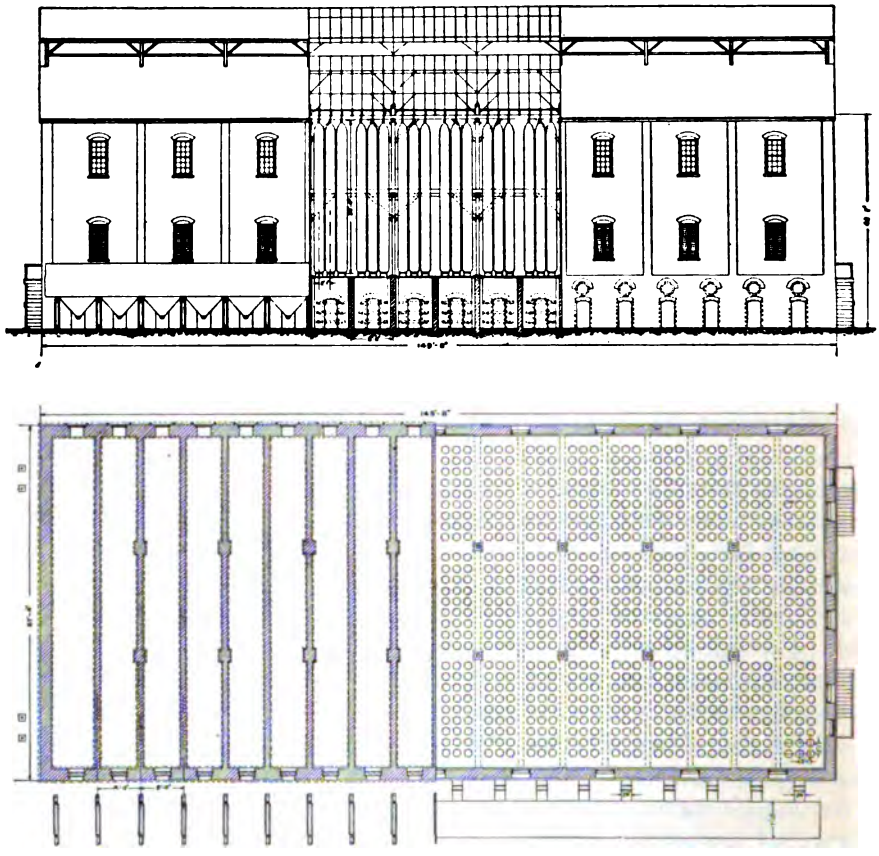


FIG. 1.—ELEVATION AND PLAN OF BAG HOUSE OF GLOBE SMELTING & REFINING CO., DENVER, COLO.

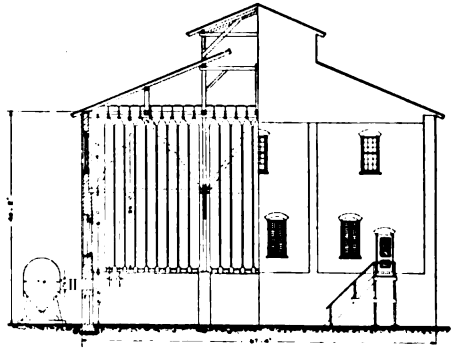


FIG. 2.—SECTIONAL END ELEVATION OF BAG HOUSE.

length of the building to allow the gas to escape. The basement is 10 ft. high and divided into compartments by brick walls 13 in. thick spaced 8 ft. apart, made tight to prevent the gases from interfering with the cleaning out of the fume in any one compartment while the others are in operation. These walls also carry the thimble floor above, which is made of No. 10 gauge sheet-iron plates riveted together as nearly gas tight as possible, so that all gas entering the basement will be forced through the thimble floor into the bags above. On the thimble floor are fastened the thimbles, which are 17 in. in diameter and 10 in. high, made of No. 14 gauge sheet iron, with a head at the top to attach the bag and a flange at the bottom for riveting to the floor plate. They are spaced on 2 ft. 1 in. centers and arranged as shown on the floor plan. To these thimbles one end of the bag, which is 31 ft. long, is fastened by means of a soft iron or

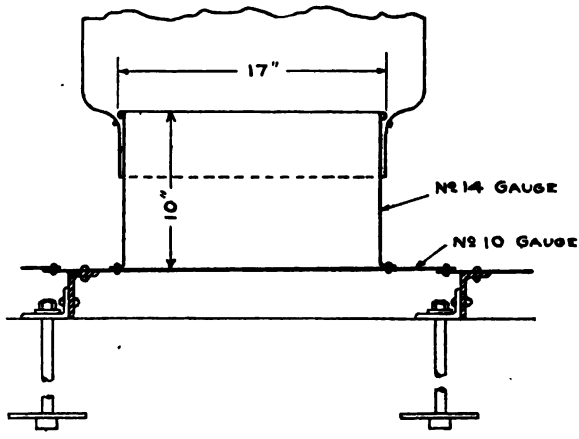


FIG. 3.—DETAIL OF THIMBLE CONSTRUCTION.

copper wire; the other end is hung by means of a similar wire fastened to the bag and then given several turns around a 2 by 12 in. plank overhead which runs parallel with and directly over each row of thimbles. These planks are carried on cross timbers resting on the frame work which supports the roof. The bags usually begin to deteriorate around the thimbles. As this takes place the lower end of the bag is cut off and the hanging wire at the top, which has been left sufficiently long for this purpose, is lengthened out; in this way the bag may still be used, although slightly decreased in filtering area. The gas is drawn from the furnaces through the flue by means of a fan and discharged into a flue passing along the side of the bag house. Short connecting flues 24 in. in diameter, equipped with gas-tight dampers, lead the gas from the main flue into each separate compartment in the basement. The thimble floor being gas tight, the fumes are forced up into the bags, the gas filtering through the

fabric and passing out through the louver on top of the roof. The fume retained in the bags is dislodged, at regular intervals, by shaking, and deposited in the basement, and is removed through the basement doors after first shutting off the gas from that compartment.

The ideal filtering material is a thin layer of absorbent cotton, but, owing to the difficulty of cleaning and recovering the fume without destroying the material for filtering purposes, it is impractical. The bags used are either cotton or woolen, and while various other substances have been experimented with, they have not been successful, either on account of excessive cost, or, lacking in nap, the material would act as a screen and fail to abstract the solid particles from the gases. Some 20 years ago the writer treated woolen cloth with titanium chloride. This treated material filtered as well as untreated cloth and resisted the corrosive effect of sulphuric acid to a point where the condensed acid fully saturated the bag. The material of the saturated bag then either clogged up and no amount of shaking would dislodge the fume and all filtering ceased, or, if the gases were low in fume and high in acid, the nap of the cloth would fold up on the strands and the fumes would pass through without filtering.

Cotton bags do not offer as great a resistance to the corrosive action of acid and will not stand as high temperatures as woolen. It is difficult to determine from the analysis of the gas which material should be used, as the acid contents and temperatures of the gases, particularly from an oxidizing furnace, are liable to vary greatly. It is better to use woolen bags whenever there is doubt. As woolen bags cost from three to four times as much as cotton, they must last correspondingly longer.

The writer recalls seeing all the bags in a building destroyed within 30 min. This happened with gases from a converter working on lead-bearing copper mattes and occurred on the finishing blow to blister, when the lead content of the converter charge was low—under 2 per cent. The converter delivered the gases into a brick flue, the opening of which made a snug joint with the converter snout. The temperature in the brick flue back of the converter ranged from 800° to 1,300° F., and at the discharge end of this brick flue the temperature ranged between 400° and 700° F. In this range, giving the proper temperature—from 800° to 1,200° F.—for conversion, by contact, of sulphur dioxide to sulphur trioxide, the amount of sulphur trioxide formed was in excess of the lead oxide carried by the gases; the surplus acid rapidly destroyed the bags. This was overcome by enlarging the opening in the flue, also an opening with a damper was made in the side of the flue close to the point where the converter discharged into it; the fan was speeded up, thus diluting and cooling the gases. A recording thermometer was placed in the flue 25 ft. from the converter, and 600° F. was the maximum reading permitted on this thermometer. The dilution of the gas was also regulated by analyses,

never allowing the sulphur dioxide content of the gases to get over 4 per cent. It had formerly run as high as 9 per cent. on the finishing blow. After these precautions were taken a set of cotton bags lasted over a year.

The same action may occur on concentrating and cupelling furnaces where the only sulphur present comes from the fuel. With the temperature of the flue as noted above, the conversion of sulphur dioxide into sulphur trioxide proceeds rapidly and destroys the bags. Fortunately, in a majority of these cases there is sufficient lead oxide in the gases to combine with the sulphur trioxide formed, rendering it harmless.

Another cause for the corrosion of bags is the presence of selenium. The selenium being volatilized and passing off with the gases as an oxide, upon coming into contact with the sulphur dioxide converts the latter into sulphur trioxide with the production of selenium, and unless there are sufficient bases to combine with the sulphur trioxide produced the bags are attacked.

The difficulties of an accurate determination for small amounts of sulphur trioxide in the presence of sulphur dioxide are well known, and while a large number of determinations were made, they were used for comparative purposes only.

Exhaustive tests were made to determine the critical temperature of cotton and woolen bags. The fabrics used in these tests were cut from the cotton and woolen bags in use in the bag house. The test pieces were cut to 3 by 4 in. and were pulled, always against the warp, in an Olsen testing machine. In making a test for any given temperature and period of time, a piece of cloth 8 by 15 in. was cut into 10 rectangular pieces, as shown below.

1	2	3	4	5
10	9	8	7	6

The pieces bearing the odd numbers were laid aside and those bearing the even numbers were placed in a Freas electric oven, which was kept at the proper temperature for the desired period of time, at the expiration of which they were removed and all 10 pieces broken in the Olsen machine. In most cases, especially at or near the critical points, two or more, and sometimes as many as five, independent tests were made. All told, something more than 1,000 test pieces were pulled. Individual tests varied, but sufficient work was done to make the results given below, which are averages, correct.

Percentage Loss in Tensile Strength Due to Heating

Cotton Cloth

Temperature of Oven, Deg. F.	Time Pieces were Kept in Oven				
	1 hr.	24 hr.	48 hr.	96 hr.	144 hr.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
210	.....	0.0	.....	.....	0.0
215	.....	0.0	0.9	6.1	9.4
220	.....	4.2	.....	.....	.....
225	.....	5.25	.....	.....	.....
230	.....	7.3	.....	.....	.....
235	.....	12.1	.....	.....	.....
240	.....	17.1	.....	.....	.....
245	.....	20.7	.....	.....	.....
260	0.3	.....	.....	.....	.....
265	3.5	.....	.....	.....	.....
270	5.3	.....	.....	.....	.....
280	9.6	.....	.....	.....	.....

Woolen Cloth

Temperature of Oven, Deg. F.	Time Pieces were Kept in Oven				
	1 hr.	24 hr.	48 hr.	96 hr.	144 hr.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
270	.....	0.0	0.0	0.0	0.0
275	.....	.....	.....	0.3	3.2
280	.....	.....	0.0	5.3	6.9
285	.....	.....	0.0	.....	.....
290	.....	0.0	2.8	.....	.....
295	.....	4.0	.....	.....	.....
300	.....	10.7	.....	.....	.....
355	0.0	.....	.....	.....	.....
365	0.0	.....	.....	.....	.....
370	2.3	.....	.....	.....	.....
375	4.4	.....	.....	.....	.....
385	6.7	.....	.....	.....	.....

Each plant has its favorite brand of cloth, which is required to have a definite number of strands; in cotton cloth these vary from 30 by 30 to 48 by 48, depending upon the character of the fume to be filtered; with woolen cloth, owing to the longer nap, a coarser weave may be used, and the number of strands is usually in the twenties. The woolen cloth should contain the natural grease, but, as the manufacturers object to this, the

wool is usually scoured and manufactured into cloth and the grease added. After numerous working tests, checked by the laboratory, the following specifications were formulated for a satisfactory woolen cloth:

The weight is to average 12 oz. per yard and the tensile strength is to be not less than 21.5 lb. per lineal inch. The test pieces are to be square, 3½ in. on a side, and the pull is to be against the warp. The fabric is to contain not less than 85 per cent. wool fiber, estimated by taking the difference between 100 per cent. and the sum of grease, dirt, moisture, burrs, and cotton fiber, and to be practically free from vegetable matter of all kinds. The weave is to be 22 ends by 20 picks per inch.

Yarn is used for sewing woolen material. Linen thread should be used for cotton bags, usually No. 40 Barbour's Irish Linen, with a double lap seam, and lock stitch is specified.

There is a diversity of opinion as to the necessity of ventilation around the bags, but the consensus of opinion is that good ventilation lengthens the life of a bag. Some plants have gone to the extent of drawing the gases from around the bags with a fan and discharging them into a stack. An iron stack 4 ft. 6 in. in diameter and 68 ft. above the roof has proved satisfactory. This stack takes care of 15,000 cu. ft. of gas per minute and maintains a draft of about 0.05 in. of water. The temperature of the gases entering this stack varies from 110° to 130° F. Diffusion stacks after the Wislicenus type were tried for converter gases. They worked nicely, but the rain beating through the openings caused rapid deterioration and they were abandoned.

The number of bags in a compartment which can be closed off from the main current varies greatly. The original Globe bag house contained 81 bags. Some smelting works have as many as a thousand bags in a compartment. As most bag houses are run continuously, one or two compartments are usually cut out for cleaning, shaking, etc., and it is therefore better that a unit be of such size as not to have too large a percentage of the filtering area cut out when one or more compartments are closed off.

The filtering area or bag surface necessary to handle a given amount of gas is entirely dependent upon conditions, and what is ample in one case may be insufficient in another. The area necessary is not only dependent upon the volume of gases, but upon the amount of solids per cubic foot, the stronger or weaker adhesion of the fume to the filtering material, and the number of times the bags are shaken in 24 hr.

Few smelters accurately measure the volume of gases being handled, but use the manufacturer's rating of the fan at the different speeds. As this depends upon character, resistance, temperature, etc., of the flue to the fan, and the same variable factors on the discharge end of the fan, the rated amount is not always delivered. Comparing performances of bag houses with fan rating, one bag house could be cited in which each bag is filtering 130 cu. ft. of gas per minute and recovering 5 lb. of fume

per bag per day, with bags shaken once in 24 hr., and another in which each bag filters 70 cu. ft. of gas per minute and recovers 27 lb. of fume per day, with bags shaken eight times during that period.

In dislodging the adhering fume from the bags, hand shaking, when done properly, gives the best results, but it is a slow and disagreeable task and is being replaced by various mechanical shakers. This mechanical shaking is accomplished by striking the inflated bag lengthwise; quickly jerking the deflated bag up and down; swinging back and forth; or a combination of the two motions. Another method is to reverse the flow of the gas through the bags by means of an individual fan or by a second connection between each compartment and the suction side of the main fan. The best results were obtained with the arrangement shown in Fig. 4. In this arrangement we have the combination of the

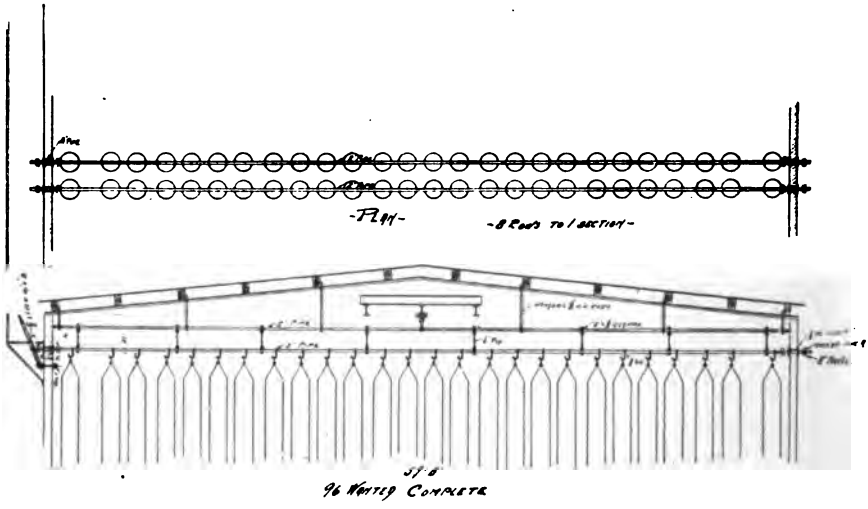


FIG. 4.—BAG-SHAKING DEVICE.

up-and-down and the back-and-forth motions. The bags in each row are hung from a 2-in. pipe, which in turn is supported from above by hangers spaced about 8 ft. apart and 21 in. long. The ends of the pipe pass through cast-iron spools placed in the walls, with holes in them sufficiently large to allow for the up-and-down motion caused by the swinging around the 21-in. radius. On the ends of this pipe are placed collars, one on each side of the cast-iron spool, and set for 8-in. stroke. A lever is fastened to one end of the pipe, which, when pulled quickly back and forth, gives the bags an up-and-down as well as a back-and-forth motion, and on coming up sharp against the collars also produces a sudden jar which assists in dislodging the fume from the bags. This device when handled properly will lower the pressure in the bags to 0.15

in. of water. Good hand shaking will lower the pressure to 0.10 in. or slightly less.

It is important that the bags are so hung that when inflated they will stand straight; otherwise, upon shaking, the fume will collect around the thimble top, distorting the bottom of the bag and thus obstructing the flow of the gases into the bag, at the same time putting an undue strain upon the bag at this point.

Fume from blast furnaces carries enough sulphides and finely divided carbon to burn. The burned material, after moistening thoroughly, can be handled safely. Sluicing the fume from the compartment promises to be the quickest and most sanitary method, but filter pressing and the large amount of moisture left in the cake are objectionable. In any case, cleanliness should be demanded from all employees and no scattering of fume should be permitted around a bag house.

Bag filtration, with its high percentage recovery of fume, diffusion of gases, and simplicity of operation, is very satisfactory, and the bag house has well proved its worth to the lead smelter. Credit for its introduction into this branch of metallurgy is due Dennis Sheedy, Manager, and Dr. M. W. Iles, Superintendent, of the Globe Smelting & Refining Co.

Effects of the Bag House on the Metallurgy of Lead

BY L. DOUGLASS ANDERSON, MIDVALE, UTAH

(Salt Lake Meeting, August, 1914)

FOR some years past the annual reviews of the metallurgy of lead have almost uniformly stated that there have been no great changes, such as there were being more particularly noticeable in the refinement of details. While this is quite true, nevertheless these changes in details are gradually bringing about a condition that, taken as a whole, constitutes a considerable advance in the art. Local circumstances have often compelled metallurgists to adopt certain peculiarities of practice which upon trial have been found advantageously applicable to more than the restricted district of origin. This paper will deal with some of the more novel features developed in recent years at the United States smelter, Midvale, Utah, particularly with reference to the influence of the bag house.

The plant is equipped with six blast furnaces, 48 by 160 in., mechanically fed; four Wedge furnaces for roasting matte and sulphide ores; four Dwight-Lloyd sintering machines; 20 converter boxes for blast roasting; storage bins for ore, fuel, and fluxes; "oxide" and "sulphide" sampling mills; bag house taking all the gases of the plant; arsenic plant for treating the dust gathered there; laboratory, shops, etc.

The most marked feature of the plant is its complete bag-house system. Of the many schemes thus far tried for preventing damage to surrounding agricultural interests the writer is aware of none as yet quite as efficient as the bag house. A common remark of visitors upon approaching the plant and viewing the clear stacks is that it appears to be shut down, whereas once inside all the evidences of industry are found.

Bag-housing blast-furnace gases presents no peculiar difficulties. The gases from properly run furnaces are comparatively cool, contain scarcely a trace of acid and do not burn the filtering medium. All that is required is a proper mechanical design of the bag house and its accessories, the provision of ample filtering area, and of convenient means for handling and treating the product. While cotton or duck bags have been used, on the whole woolen bags woven of the raw wool appear to give sufficiently longer life to warrant their heavier expense. As for the filtering

area required, no definite rule can be laid down, as it is dependent upon the amount of and physical character of the fume carried in the gases and the frequency and efficiency of shaking. However, from 0.3 to 1.5 cu. ft. per minute have been filtered per square foot of area supplied, with 0.5 to 0.7 cu. ft. per minute the more usual quantities. The pressure carried is usually from 1 to 2 in. of water. Higher pressures can be carried, with consequent greater filtering capacity, but at the expense of more power and a somewhat shorter life of the bags.

The dust in the Midvale bag house is caught in V-shaped steel hoppers, through the bottoms of which run 9-in. screw conveyors by means of which the dust is regularly removed into tram cars which carry it to the arsenic plant for treatment. This idea, which was developed here, has resulted in much healthier conditions for the attendants and has been applied since to other bag houses, some of which use no hoppers but let the dust form its own slope in closed cellars down to the screw. How successful this is cannot be definitely stated, but as most dust tends to "hang up," requiring some rapping of the steel hoppers to bring it along, there is the possibility that screw conveyors may not be entirely effective when applied in rectangular cellars. It is to be noted that screw conveyors as ordinarily used run in sèmicircular or square boxes. The natural extension of the idea to dust hoppers would be to simply attach the boxes to the hoppers as in *A* or *B* of Fig. 1. With bag-house dust, however, such arrangements will invariably choke up and stop the worms. A better arrangement is shown in section at *C*. This arrangement seems to give the dust a chance to relieve itself on the sides instead of choking and breaking the screws. The present manner of shaking the bags was developed some years ago at this plant in an effort to save expense in the replacement of torn bags. It consists simply of cutting off the blast from any one hopper and applying a slight vacuum, repeating the operation two or three times. It is not as efficient as hand shaking or shaking by mechanical means. It is, however, healthier than hand shaking and, when done often enough (every 8 hr. at Midvale), is sufficient. Furthermore, bags often reach a peculiar state of fragility, when they are still efficient filtering mediums but so weak that they tear if handled roughly. It can be readily understood that gentle alternate deflation and inflation in place of rough shaking will greatly prolong their life when in this condition.

In contrast to blast-furnace gases, those from the roasting operations present many difficulties, due chiefly to their heat and the fact that they carry appreciable quantities of sulphur trioxide (SO_3). In cooling the gases sufficiently to avoid charring the bags this sulphur trioxide unites with the water vapor and becomes exceedingly destructive to the bags, whether of cotton or of wool. Under some conditions of roasting, where the ores are high in lead or zinc, the fumes from these metals unite with

this acid and render it inert. Until the invention of the Sprague process of neutralizing this acid[1] the majority of lead smelters were unable to bag-house their roaster gases. As a matter of fact, blast-furnace gases carrying practically no acid should not be nearly as harmful to vegetation as those from roasting operations, which are so hot as to permit little settling of lead and arsenic fumes and usually carry noticeable quantities of sulphuric acid. The neutralization of this acid effects two things,

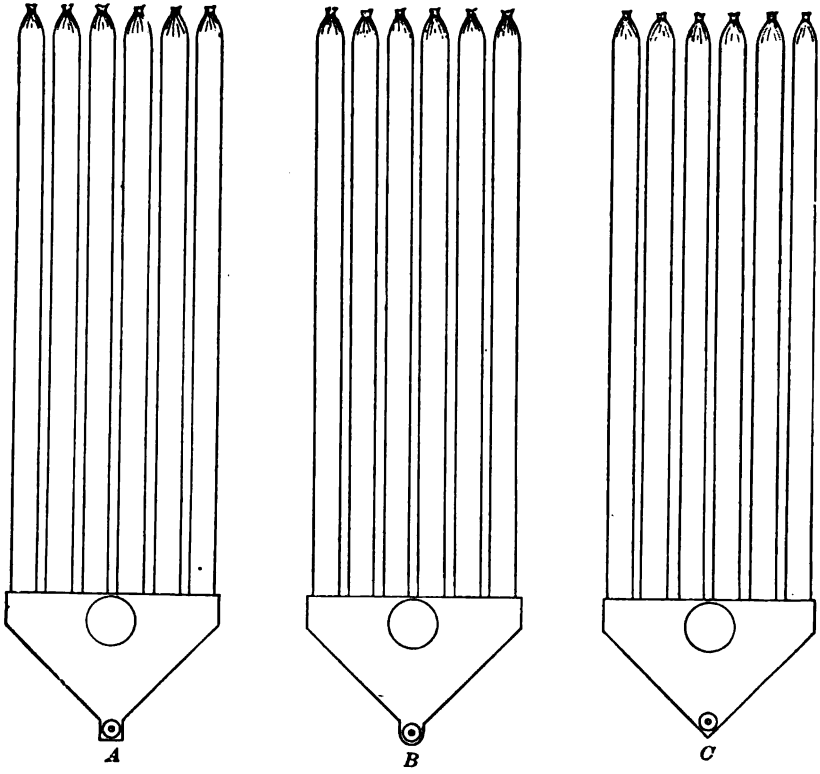


FIG. 1.—DIFFERENT ARRANGEMENTS OF HOPPERS AND CONVEYOR BOXES.

permitting filtration through woolen bags and nullifying the harmful effect of the acid on vegetation.

In the matter of metallurgical recovery the bag housing of roaster gases is also more important than that of furnace gases. In his paper on the bag house before the Institute in 1912,[2] Anton Eilers gave figures which showed that the operation of the bag house on furnace gases involved an actual loss financially when all items were taken into account. This should prove true in all plants where the furnaces carry a small

[1] U. S. patents Nos. 931515 and 992391.

[2] Notes on Bag-Filtration Plants, *Trans.*, xliv, 708 to 735 (1912).

amount of lead on the charge, say 8 to 12 per cent., and are run with properly cool tops. In such cases extensive settling chambers will save all values that are economically worth while. With more lead on the charge, however, and when agricultural interests must be protected the bag house is well worth while. But from the standpoint of metallurgy the roaster gases are those which will pay the largest returns on the investment. If hot tops in blast-furnace work are the cause of metal losses it is just as certain that the heat engendered in roasting likewise produces metal losses. This was indeed the primary reason for abandoning the old "fusion-box" type of roaster. At Midvale the fume collected in the bag house from the roaster gases carries 35 to 42 per cent. lead and about 10 oz. silver per ton, whereas that from the furnace gases carries only 18 to 22 per cent. lead and less than 1 oz. of silver per ton.

In his article on the progress of the metallurgy of lead in the first volume of *The Mineral Industry*, published in 1893, Professor Hofman called attention to the increasing exhaustion of the oxidized and carbonate ores of lead and the necessity of handling more of the sulphide ores, which were developed as the mines increased in depth. Roasting is a practically indispensable part of the treatment of these sulphide ores. In his paper on the bag house already quoted Anton Eilers mentioned the increased metallurgical losses due to the greater amount of roasting now being done. In this connection may be mentioned some experiences with the Dwight-Lloyd sintering machines, which have done so much for the whole industry. When originally installed at Midvale they were equipped with gasoline burners for ignition, which were somewhat troublesome and erratic in action. Some bad slags were produced in the blast furnaces when handling their product, which were finally traced to green sulphides carrying zinc blende coming over unignited with other parts of the charge which had properly sintered. Screening the roast was resorted to, with beneficial results. Finally a muffle type of burner using a low-grade gas oil, understood to have been developed first at Monterey, was applied. A good solid sinter was obtained without green spots and screening was abandoned; indeed, had to be, or otherwise the furnace charge became too coarse for good reduction. But the terrific heat developed by the new burners, and apparently necessary for good ignition, caused an increased loss of lead in the sintering operation. Many smelters throughout the country have lately begun to realize this. Yet the Dwight-Lloyd machine is too valuable to be impeded in its progress by this development. A more gentle ignition may cut down these metal losses. Nevertheless if they would be prevented to the maximum extent possible the bag house remains the final resort, not only for the gases from these machines, but for those from all lead-roasting operations.

It may be claimed that sufficient flues and settling chambers should throw down practically all the As_2O_3 and SO_3, but it is not recalled that

any lead smelter in this country has such an extensive system ending in a stack as to prevent a dense cloud escaping from that stack, unless all the fume is filtered.

Aside from the acid nature of the roaster gases, overcome by the Sprague process of neutralization, the cooling of these gases has been the most serious problem in bag housing them. Many efforts have been made to produce filtering fabrics which would withstand both heat and acidity, such as by chemically treating cotton and wool, using asbestos, etc. These efforts are to some extent misdirected, for effectual filtering cannot be accomplished until the gases are cooled sufficiently to get the fumes into a condition more nearly approximating what is considered solid matter, nor can the gases be so cold that condensation of moisture occurs to destroy the filter fabric. Any attempt to filter hot gases will be disappointing, resulting in visible clouds of smoke as the gases passing through the bags meet the outer air, become cooled, and precipitate the fumes which had escaped the bags in a fine state of division. As long as the gases are hot the minute particles of fume seem to repel each other and exist in a sort of smoky cloud. Cooled, however, they tend to coalesce and thereby reach a state wherein they are easily retained in the meshes of the bags. Those who have given any study at all to the matter are well aware of the distinction between true flue dust and fume. The former can be quite readily settled out of the gases by large chambers, centrifugal devices, or wet washers. But lead-smelter gases carry as their chief solids, in addition to flue dust, various metals, principally lead, zinc, and arsenic and their compounds, in a state of extreme fineness, popularly mistaken for volatilization. The failure to recognize the distinction between flue dust and fume has resulted in many failures in efforts to apply to smelter gases wet washers or centrifugal devices which were quite effective on the gases from iron blast furnaces. What success has been attained with wet washers has been chiefly due to the cooling action effected. The importance of cooling cannot, therefore, be overestimated.

Two difficulties brought on by simple cooling in flues are overcome by the bag-house system as now developed. The poor draft in stacks resulting from cool gases is taken care of by the forced draft used. The precipitation of water vapor and its combination with SO_3 to form actual H_2SO_4 is overcome by neutralization. The effectiveness of this neutralization is illustrated by measures taken to preserve certain steel cooling flues. Originally most of the neutralizer was admitted to the flue system where it would be most effective; that is to say, some distance from the roasters, where the gases were cooler. But cooling had proceeded far enough between these two points to cause the gases to seriously corrode the steel plates. Finally a certain amount of slaked lime was fed into the gases at the beginning as well as at the end of the cooling flues. From that time on internal corrosion ceased entirely.

Three methods of cooling are commonly in use: by dilution with external air, by passing through thin metal flues, and by spraying with water. The first method has only a limited application, lest the quantity of gas handled become excessive. The second method is probably the more extensively used. The rate of heat transmission through sheet-steel plates about $\frac{1}{8}$ in. thick under smelter conditions appears to be about 0.02 to 0.04 B.t.u. per minute per square foot of cooling surface per degree Fahrenheit difference between interior and exterior temperatures, depending upon cleanliness of the flues, convection by winds, shading from direct rays of the sun, etc. In computing the required area of a flue for cooling it is not sufficient to assume the mean interior temperature as the arithmetical mean of the initial and final temperatures, for the drop in temperature proceeds more slowly as the gases become cooler. If t_1 be the initial temperature of the interior gases, t_2 the final temperature, and t_s the temperature of the external air, the mean temperature difference θ_m between inside and outside will be

$$\theta_m = \frac{t_1 - t_2}{\log \frac{t_1 - t_s}{t_2 - t_s}}$$

In the use of sheet-metal flues it has been found by experiment that the cooling, within certain limits, is proportional to the square root of the velocity. For this reason it is advisable to keep the velocity high in such flues, which also tends to avoid too heavy a deposition of the heat-insulating dust.

Cooling by water sprays has been used to some extent and will undoubtedly be extended now that efficient atomizing nozzles are obtainable. Care must be used, however, that too much water be not used and the saturation point of the gases be not too closely approached, as any dampness of bags is fatal to their life. It is an interesting fact that by full utilization of the latent heat of evaporization gases can be cooled by warm water below the temperature of the water itself. Those who are interested in this subject are referred to a paper by Willis H. Carrier, entitled Rational Psychrometric Formulæ.[3]

In the Midvale bag house, roaster and blast-furnace gases and dusts are kept entirely separate. The roaster dust, carrying 9 to 16 per cent. arsenic, is briquetted with other flue dust and fine ores. The lime fed in as a neutralizing agent provides an efficient binder. These briquettes are then charged to the blast furnaces. The dust from the blast-furnace section of the bag house carries from 35 to 45 per cent. arsenic. This dust is treated in the arsenic plant, where by two successive volatilizations a high-grade fine white commercial arsenic product is made, running better

[3] *Transactions of the American Society of Mechanical Engineers*, vol. xxxiii, pp. 1005 to 1039 (1911).

than 99.5 per cent. As_2O_3. It is seen, therefore, that there is a gradual concentration of the arsenic through the successive steps of the roasters, the bag house, the briquetting press, the blast furnaces, the bag house again, and finally the arsenic plant.

The details of arsenic manufacture were ably treated last year before the Institute by James O. Elton. Further comment must be restricted to mentioning the troubles experienced when the blast-furnace bag-house dust approaches 45 per cent. in arsenic content. At all times it is an inflammable material. The success of the hoppers with their screw conveyors is due to their being kept cool by free air circulation around them and regular frequent removal of the dust. If this were allowed to accumulate in large quantities it would catch fire and burn to crusts, which would destroy the revolving screws. Furthermore, the arsenic content must be kept under 45 per cent. or the dust will tend to burn in the cars on the way to the arsenic plant, giving off heavy white fumes, making it well-nigh impossible to handle. For this reason the metallurgist in a plant of this kind receiving many ores high in arsenides has to watch his charges for arsenic in addition to the usual elements.

Having caught the material which formerly escaped from the stacks, his problem is to extract from it the valuable metals and get the arsenic out of the system as quickly as possible, minimizing any closed circuits. Such a one, for example, is formed in the briquetting of the arsenic-plant residue or sinter after roasting. What arsenic was not roasted out of the bag-house dust on the Brunton hearths goes back to the blast furnaces, then to the bag house, and from there once more to the arsenic plant. It will also be found that certain of the less common elements will tend to build up in the sinter unless it be eventually removed from the system for further treatment, cadmium being especially noticeable, in addition to small quantities of tellurium, selenium, etc. All told, then, it will be seen that the bag-house system introduces some interesting new problems that the lead metallurgist often neglects to consider.

It is worthy of note that the bag house should not be depended upon as a "cure all" for bad furnace running. No metallurgist should assume that because he has a bag house to catch the fume he can be reckless about the condition of his furnace tops with the idea that the bag house will catch the values which may be given off. In the first place, hot tops will produce hot gases, which will be hard on the bags. In the second place, even if the lead fume is arrested in the bag house it has to be treated again at heavy expense, and with a second loss due to handling, slag losses, etc. The blast furnaces should be run fully as carefully with a bag house as without, if the maximum recovery would be obtained. To get the maximum percentage of the lead into the bullion with a minimum output of by-products is still the rule for high recovery.

Where all the gases of a lead smelter are put through a bag house it

will be found that most of the lead loss can be traced directly to the slags, largely simplifying the metallurgist's problem. It must be borne in mind that this applies only when all the gases from every operation whatsoever are filtered. The slag-settling furnace has proved of considerable aid in cleansing the slags finally rejected. Its actual saving, however, cannot be figured from the difference between the lead in the slag fed to it and the lead in the slag thrown away. A considerable amount of lead must escape from the stack together with the products of combustion. It is on record that certain metallurgical plants were able to make wonderful records for clean slags, only to find after a year's campaign that there was a distressing and mysterious large percentage loss. Such experiences as these are forcing metallurgists to recognize the fact that lead is rather an elusive metal and that it has a very large avenue of escape in the fumes which have been poured out so unrestrainedly in the past.

The installation of four Dwight-Lloyd sintering machines in this plant introduced an unforeseen complication in connection with the bag-house system. Soon after they were gotten into operation mysterious sudden rises of temperature and pressure in the flue system began to occur. The dust caught in the bags changed in character, becoming sticky and clogging them up so much as to require vigorous hand shaking to free them. Finally one of the aforementioned sudden rises in temperature became so great as to set fire to several hundred bags, the dust in the hoppers and flues also burning vigorously. It began to appear as if the gases from the Dwight-Lloyd machines could not be filtered. As the plant could not run unless all gases went through the bag house, extensive experiments were set under way in an effort to solve the difficulty. It finally became clear that the trouble was due to the production of free elemental sulphur, which occurred in the gases in a state of vapor. This free sulphur vapor would apparently ignite on small provocation, there appearing at times to be an actual spontaneous combustion of the material with the arsenic present, considerable arsenious sulphide being produced, while at other times probably sparks drawn through the wind boxes of the sintering machines were the cause of ignition. The steam generated from the moisture contained in the sintering-machine charges probably produced the neutral or reducing atmosphere required in the Hall process of producing free sulphur. Here, however, this free sulphur was most unwelcome and every effort was made to oxidize it or arrest it before it reached the bag house, but with very small success. Water sprays of many types were tried, but the free sulphur vapor proved practically as elusive as lead fume, which all metallurgists know is scarcely touched by water. Strangely enough, the percentage of free sulphur produced became less and less as the weather became warmer and the air drier. Probably if the machines could be supplied with dry air, after the manner of the pyrites burners in the Mannheim contact process for the manu-

facture of sulphuric acid, the production of free sulphur could be prevented. The cost, however, appeared prohibitive. Among many other schemes tried was the use of crushed limestone on the grates under the ore charge proper. This did to a certain small extent diminish the percentage of free sulphur produced. It was not sufficient, however, to prevent a second explosion in November, 1913, when the atmospheric air had again become damp. When this occurred the machines were shut down except for sintering preroasted matte, which operation did not appear to generate free sulphur, and for sintering sulphide ores low in pyrite.

"Conservation of resources" is in the air everywhere. A great contribution to this conservation is the closer saving of the raw materials being made in all branches of industry. The lead smelters have not yet reached the perfection of saving made by the packers who boasted of using all parts of the pig except the squeal—and are accused of utilizing even that now that they are under investigation by the Federal government! Yet progress is being made. The increasing development of hydrometallurgy may afford a market for the acid which could be made from the sulphur; even the iron of the slags may be recovered in such processes. In this progress a most important contribution has been made by such men as Percy, one of the first to suggest filtering mediums for smelter fumes; Lewis and Bartlett, whose successful bag process was the inspiration of many bag houses; the able and distinguished technical staff of the American Smelting & Refining Co., who solved many troublesome details; Robert D. Rhodes, whose patience and determination brought roaster gases within the province of the bag house; Sprague, whose neutralization process removed a large stumbling block, and many others whose names time does not permit us to recall.

Lead-Matte Converting at Tooele

BY OSCAR M. KUCHS, TOOELE, UTAH

(Salt Lake Meeting, August, 1914)

IN January and February of the current year, at the Tooele plant of the International Smelting Co., Tooele, Utah, a modification of the usual method of converting, for the treatment of copper-lead matte from lead blast furnaces, was developed and put into successful operation. In brief, the method consists in blowing molten matte direct from the blast furnaces in basic-lined converters, without the addition of siliceous fluxing ores, to the almost complete elimination of the lead contained, and to the production of copper bullion, the resulting lead fume being recovered by the filtration of the converter gases through woolen fume bags.

The process is carried out in conjunction with the converting of reverberatory copper matte, the heavy iron slag, together with the copper precipitated, being transferred to converters blowing copper matte. The purpose of this transfer is to reduce the copper oxide formed, due to the necessary overblowing of the lead-matte charge, and to utilize, as far as possible, the value of iron from the lead matte, for the fluxing of siliceous dry ores. The final slag from the copper converters is then skimmed off and transferred molten to a reverberatory smelting furnace treating copper ores.

This variation from general converting practice was the outgrowth of results obtained during 1913 when it was endeavored to convert copper-lead matte in the usual manner, with the direct addition of siliceous fluxing ores. To make clear the reasons lying back of the process it is necessary to recount at some length the results of operations during that year.

In the design of the lead plant it was planned to treat the copper-lead matte from the furnaces by direct converting without reconcentration and thereby to recover the copper, silver, and gold as quickly as possible. A converter plant of ample capacity, treating copper matte, already being in operation, it was only necessary to install a bag house, a fan, and the necessary flues to complete the equipment.

The bag house constructed has a bag-cloth filtering area of 136,000 sq. ft., provided by 960 fume bags, 18 in. in diameter by 30 ft. in length. A No. 20 Sirocco fan of 180,000 cu. ft. capacity delivers the gases from

the converter plant to the bag house. The converter plant consists of five stands of horizontal, cylindrical type shells 96 by 150 in. in size. A common flue received the gases from all stands. The admission of outside air was mainly relied upon for cooling the gases to the desired temperature. Means were provided for by-passing gases to the stack in case of excessive temperatures. Temperature control of gases was made possible by the installation of recording thermometers.

Converting operations were begun in February, 1913, with the bag house equipped with woolen fume bags of cross-woven fabric, 24 threads to the inch, warp and filling. As regards the converting operation itself, test runs had shown that no difficulties attended the handling of this type of matte and operations were conducted in a manner parallel to the treatment of copper matte.

For a period of two months no difficulties were encountered. The production of by-products, however, appeared as a disadvantageous feature. The passage of 45 per cent. of the lead contained in the matte, to the slag, necessitated its re-treatment in the blast furnaces.

Bag-house difficulties began after several months of operation. Leakage developed first through perforations at the tops of the bags. While leakage of water through a prepared roofing material, that had become affected by the converter gases, was partly responsible for this condition, it soon became apparent that the destruction of the fabric was due to condensation of acid inside the bags. This condensation manifested itself in the form of spots, first at the top and later at the bottom of the bags, about 5 to 6 ft. at each end being affected. At the points where it appeared the fabric became weakened by acid corrosion and would eventually be blown out by fan pressure, leaving enlarged perforations. The nap of the cloth was also rapidly corroded away, giving a screen-like appearance to the bags, and finally a general weakening of the whole fabric resulted in their being torn from the thimble floor. Frequent renewals of bags became necessary to prevent leakages, and inability to correct the underlying causes made it necessary to discontinue operations pending a thorough investigation of the problem.

A solution of all the difficulties required either the destruction of acid by the addition of a neutralizing agent, such as ZnO or CaO, or the production of a fume in itself sufficiently alkaline to preclude the formation of acid, and the production of a converter slag sufficiently low in lead to permit of its disposal by pouring back into reverberatories.

As a result of considerable experimental work, the process as outlined at the beginning of this paper was worked out. Tests carried out intermittently during the day shift in a coverter set aside for the purpose showed a rapid formation of a uniform magnetite coating on the walls of the converter. It was expected that difficulty would be encountered due to the gradual closing in and reduction in capacity of the converter.

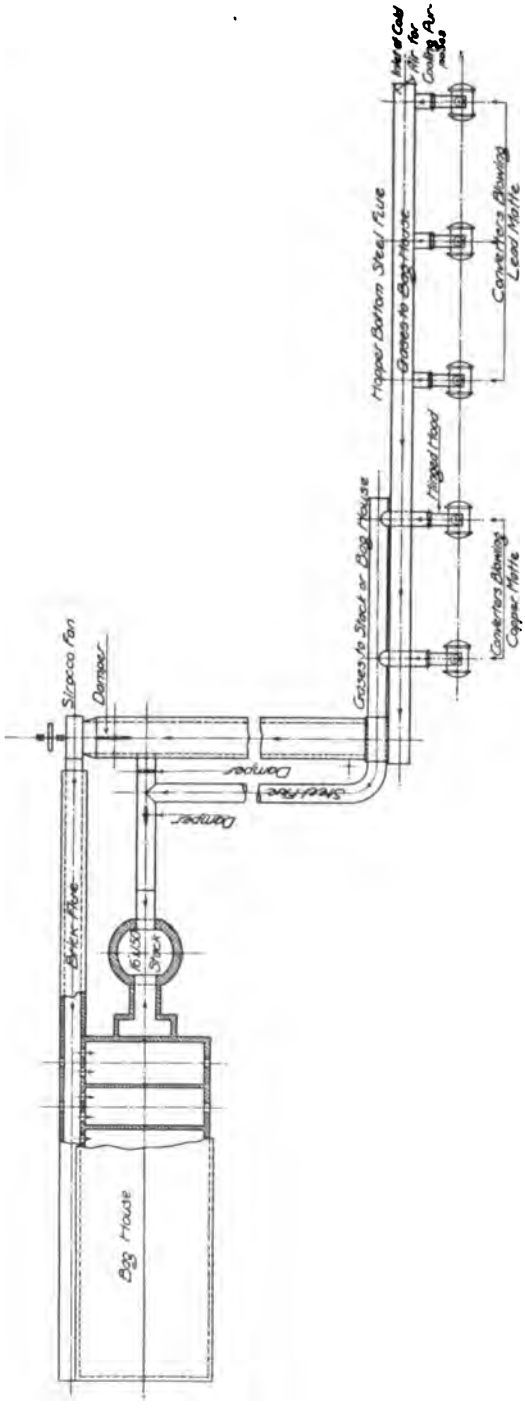


FIG. 1.—ARRANGEMENT OF CONVERTERS, FLUES, AND BAG HOUSE FOR BLOWING LEAD MATTE.

This, however, under continuous operation did not develop, the crusting under these conditions being confined to the wall opposite the tuyères, being heaviest at the mouth, due to spattering of the charge, and thinning out toward the bottom. The fume produced was more actively alkaline, the percentage of zinc eliminated from the converter as ZnO approximating 50 per cent., as against less than 10 per cent. under the previous conditions. It was not deemed necessary, therefore, to consider the addition of a neutralizing agent to the gases.

Preparatory to putting the process into continuous operation, changes were made in the flue system in order that gases from lead converters only would be filtered through the bag house. Three converter stands were consequently connected with a separate flue for lead-matte work, the remaining two stands being connected directly to the stack for copper-matte blowing. Fig. 1 shows the present arrangement. The necessity for better temperature control was realized and flues were equipped with dampers to control the admission of outside cooling air. An automatic alarm system was installed, to record by gongs and colored lamps the passing of the allowable maximum and minimum temperatures. The operating range of temperature of the gases entering the bag house is maintained at from 210° F. to 230° F. The bag house was equipped with bags from the La Porte Woolen Mills, Pendleton Woolen Mills, and Knight Woolen Mills. The former are cross-woven and have 24 threads to the inch, warp and filling; the latter is a twilled weave bag.

Converting operations were begun in February, 1914. The formation of a monolithic magnetite lining, as stated, did not develop. Rather rapid building up of the converter mouth at first gave considerable trouble; however, enlargement of the mouth relieved this condition materially. Corrosion at the tuyère line proved to be somewhat more severe than results from the blowing of copper matte, and more frequent patching is necessary.

The time required for the blowing of a charge of 10 tons is about 2 hr., the time depending on the care given to the tuyères. Frequent punching is necessary. A tendency on the part of the tuyères to blind when silica is absent is very marked. Fume is evolved in dense clouds as soon as blowing begins. The volume of fume gradually diminishes after the blow is half over, and finally, if the blowing is continued long enough, fades out. The end of the blow is thus indicated by the character of the fume and flame discharged from the converter, the flame becoming more and more visible as the fume diminishes. The charge in the converter at this stage contains about 1 to 1.5 per cent. lead and has been reduced in weight to approximately 65 to 70 per cent. of the matte charged. The fume produced under the new conditions does not seem to affect the bags materially, conditions at the bag house being greatly improved. After three months of operation only four bags had failed.

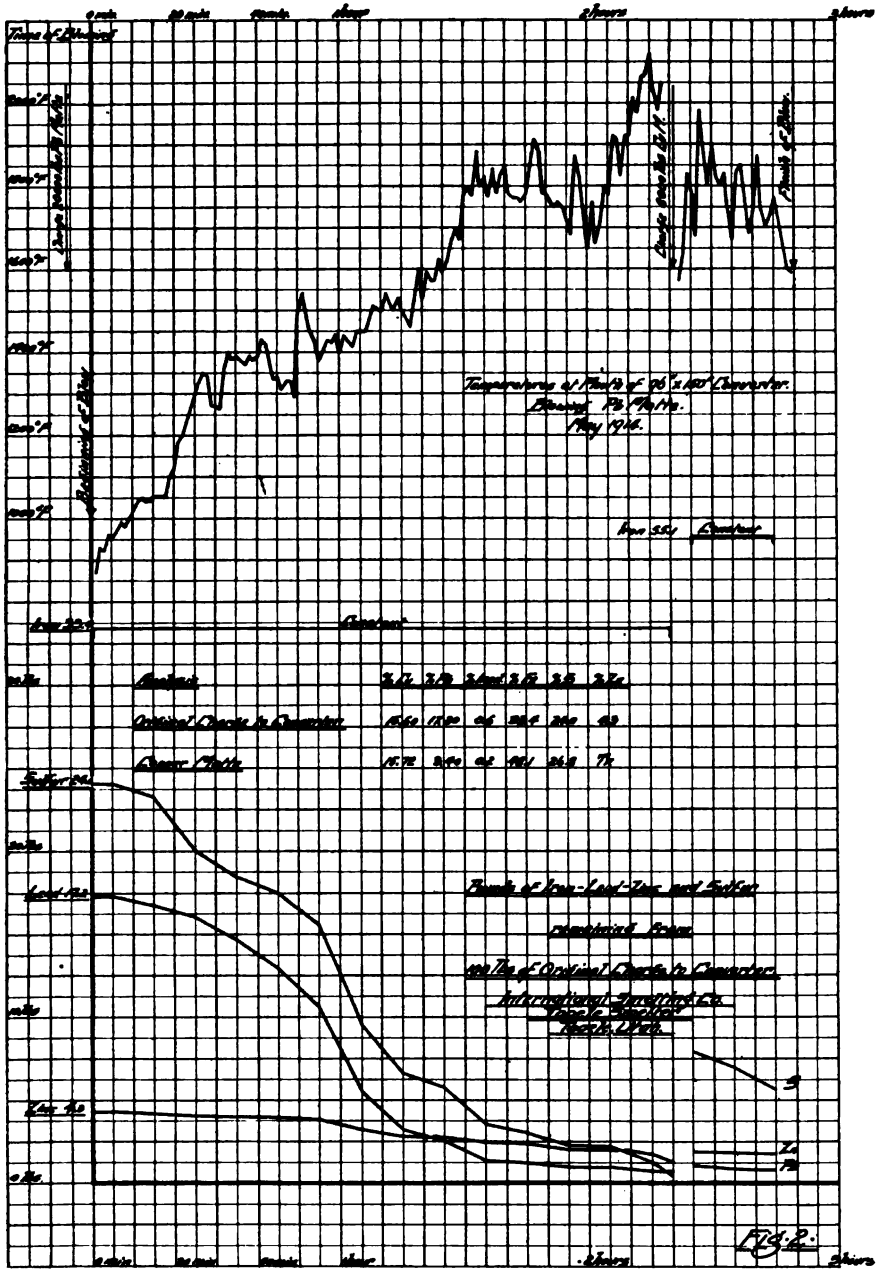


FIG. 2.—TEMPERATURE AND ELIMINATION CURVES.

These failures were due to apparently defective material, in that one failed through splitting at the seam and the remainder were torn, shaking of the bags appearing to have gradually opened up and extended flaws in the fabric. No acid spotting of the bags is noticeable.

To show the rate of elimination of lead, zinc, and sulphur from matte and the temperatures generated when the blow is continuous and the tuyères are kept fairly free, elimination and temperature curves applying to a single test are given in Fig. 2. These curves are derived from data obtained during a test when copper matte was added at the end of the blow to reduce copper oxide. Iron is taken to be constant in the converter. These curves show about 75 per cent. of the lead and 65 per cent. of the sulphur are eliminated in 70 min. of a 140-min. blow. Zinc is eliminated at a fairly uniform rate throughout the blow.

The following are comparative figures of matte blown and products formed under the former practice as compared with the present practice:

1913—*Converting Lead Matte with the Addition of Siliceous Fluxing Ores*

	Pb, Per Cent.	Cu, Per Cent.	Ag, Oz. Ton.	SiO_2, Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Zn, Per Cent.
Matte	16.5	7.66	32.8	.....	37.4	.....	22.5	5.0
Siliceous ore	3.4	0.3	21.3	62.2	6.0	5.1	0.7	.....
Converter slag	6.8	5.8	9.2	22.0	38.7	.....	3.2	.....
Bag-house fume	63.3	1.2	5.6	1.1	.....	.....	67.5	2.5

1914—*Converting Lead Matte without the Addition of Siliceous Fluxing Ores to Lead-Matte Charge*

Lead Elimination Blow:

	Pb, Per Cent.	Cu, Per Cent.	Ag, Oz. Ton.	SiO_2, Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Zn, Per Cent.
Lead matte	15.0	9.05	20.3	.....	37.9	.....	23.0	5.4
Transfer slag to copper converter	1.4	.....	.....	1.7	59.1	.....	1.8	4.0
Bag-house fumes	64.2	0.37	5.7	0.2	0.4	.....	6.00	10.4

Copper Converter Blow:

Copper matte	....	22.2	21.8	....	41.7	....	26.6	....
Siliceous ore	3.4	0.3	17.1	75.1	7.4	3.0	0.8	....
Converter slag	2.3	1.79	1.6	23.4	48.8	1.5	1.2	2.9

As indicated by these analyses, the converter slag from present practice can be disposed of through either reverberatories or blast furnaces, depending on economic conditions. The high active alkalinity of the fume, as indicated by the percentage of ZnO present, should afford ample protection against acid corrosion for the fume filter bags, the condition of the bag house after nearly four months of operation seeming to vindicate this view.

In conclusion, I desire to express my appreciation of assistance rendered by A. Austin, J. C. Welch, and A. C. Wandel in the preparation of this paper.

Basic-Lined Converter Practice at the Old Dominion Plant

BY L. O. HOWARD, GLOBE, ARIZ.

(Salt Lake Meeting, August, 1914)

THE practice of using acid-lined converters at the plant of the Old Dominion Copper Mining & Smelting Co. was discontinued early in January, 1913, and replaced by basic-lined converters. The new equipment consists of one electrically operated 12-ft. upright converter stand and two basic-lined shells for same. These shells are provided with twenty-four 1 1/4-in. tuyères. The mouths are 5 ft. in diameter.

Owing to the fact that there is no bedding system or other large storage capacity between the mine and the smelter, the furnaces must handle as fast as produced ore and concentrates, which may vary considerably from day to day, both in tonnage and copper content. Consequently the converter is often crowded to its utmost capacity by the frequent fluctuations in the daily matte supply. From the foregoing, it will be seen that in no way can the converter be favored by running light charges or by transferring to it from other stands in operation. Neither can it regularly be allowed to cool between blows, and only occasionally can it be run at as low a temperature as might be wished. No copper precipitate or other foreign material high in copper is available to help out the tonnage. Repairs and alterations or the removal of defective tuyères must wait until such time as a slight falling off in the matte supply allows the converting operations to ease up. These variable conditions all combine to interfere with the systematic operating of the converter.

The first 12-ft. shell was blown in on Jan. 6, 1913, and ran until June 27, 1913, when it was removed to patch the brick. A record of operations covering this period follows:

Shell No. 1. *First Campaign, Jan.* 6 *to June* 27, 1913

Commenced blowing	Jan. 6, 1913.
Taken off stand for patching	June 27, 1913.
Total blowing time	2,689 hr. 40 min.
Total number of blows made	664
Average time of blow	4 hr. 3 min.
Tons matte charged	18,157
Average copper content of matte	43.9 per cent.
Average time blowing one ton copper	22.5 min.
Total copper made	7,250 tons

Average copper per converter hour	2.66 tons
Average air used per minute	9,000 cu. ft.
Average blast pressure	13.3 lb.
Ore fed	3,304 tons

Average Slag Analysis	Per Cent.	Average Matte Analysis	Per Cent.
Cu	2.5	Cu	43.9
SiO_2	22.7	Fe	28.8
Fe	52.1	S	22.6
CaO	0.8	Ins	0.4
Al_2O_3	1.6		

The average air—9,000 cu. ft. delivered by the blowing engine—is calculated from cylinder displacement multiplied by revolutions for the given time, assuming 100 per cent. efficiency for the engine, and is in excess of that used in converting, for, as only one converter is in operation, some air must necessarily be wasted whenever the shell is turned down while skimming, charging, etc.

Between Jan. 6 and Feb. 15, 1913, the casting machine was not in commission and copper was poured from the converter directly into molds. This operation took about 1 hr. per blow, and the molten charge cooling in the front of the converter mouth caused it to close up rapidly. To remove the incrustations thus formed, hooks of various sizes and shapes were made, but none proved satisfactory and the opening gradually grew smaller until it measured only 18 by 24 in. Operations were then suspended for 24 hr. and 14 concentric holes parallel to the sides of the shell were burned through the accumulation to a depth of 4 ft. with an electric arc. Using round iron wedges and driving them into these holes by pounding with a heavy weight suspended from the crane, it was possible to break up the mass and enlarge the opening to its original size. The cause had not been overcome, however, and this condition was a constant source of annoyance until the collar puller shown in Fig. 1 was designed. By its use the opening in the mouth of the converter can be kept at any desired size. It works rapidly and very effectively, easily cutting through hot incrustations 3 ft. in thickness. Unlike a hook, it does not tend to suddenly pull loose, and therefore if the shell has accidentally been lifted off the rolls in the efforts to remove the collar there is less chance of its dropping and breaking the pillow blocks. In use, this machine, suspended from the crane, is lowered into the converter and the cutting tool brought up against that part of the collar to be removed. On lifting, the "tail" comes up against the opposite side of the shell, and not only prevents the tool from pulling loose but pushes the cutter into the collar. If the incrustation is very heavy, deep parallel channels may be cut that will ultimately weaken it sufficiently to allow

large pieces to be pulled loose. After the opening has been enlarged to the desired size the mouth is coated with mud, which forms a line of weakness that renders subsequent collar pulling less difficult. The use of a casting machine has, of course, made pouring copper into the molds from the converter unnecessary and since this practice has been discontinued the mouth does not build up as in the past.

As no one in the plant had any previous knowledge of basic converting, the mistake was at first made of attempting to blow very large charges. Trouble immediately resulted. Tuyères frequently became blocked and had to be replaced and in so doing the lining was injured,

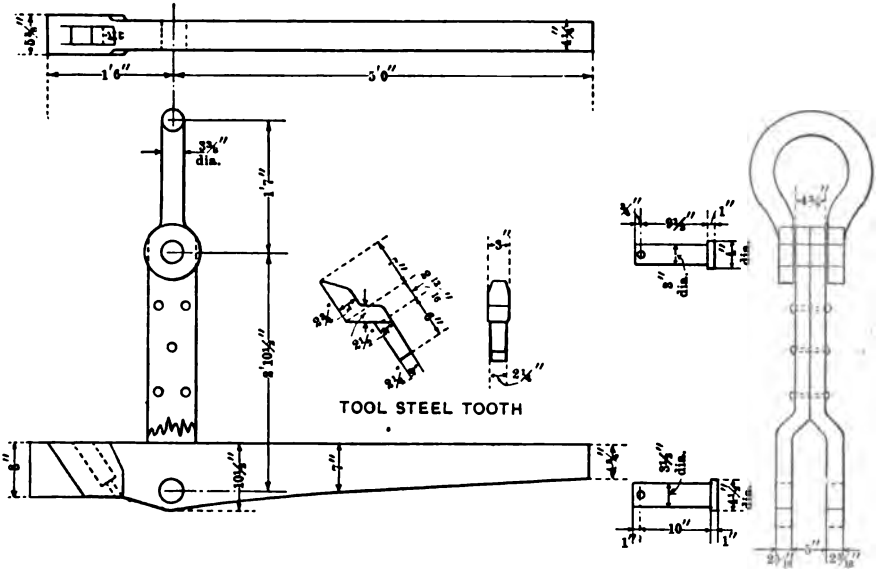
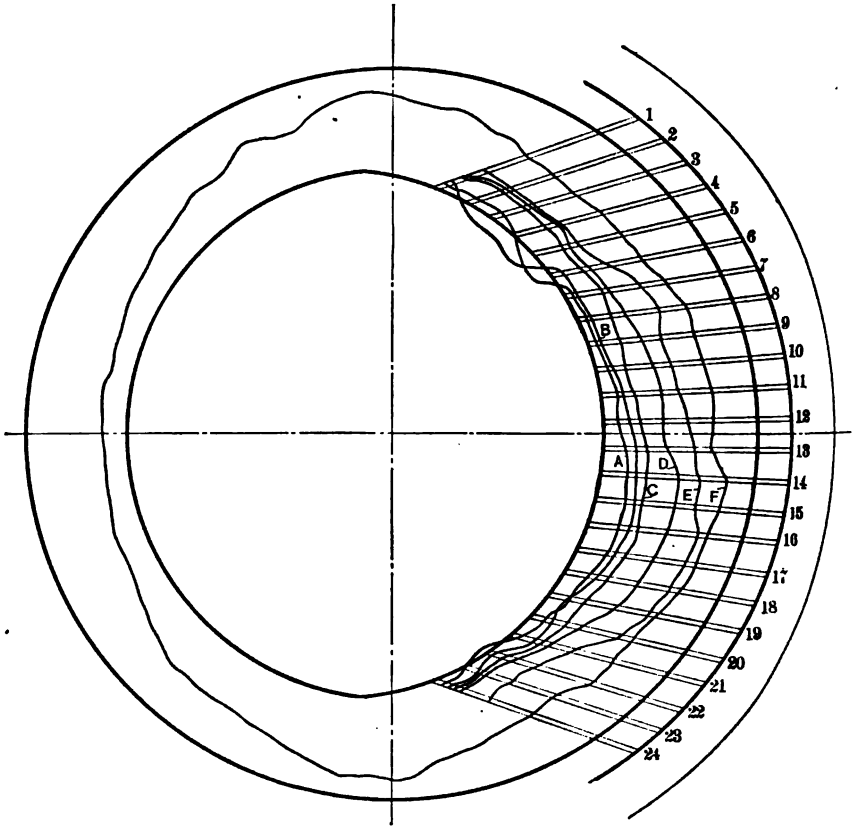


FIG. 1.—COLLAR PULLER.

notably between tuyères 13 and 17. Fig. 2 shows that the wear at this point kept in advance of the remainder of the shell throughout the campaign.

To guard against any possible slagging of the brick by silica, a slag more basic than had been the custom in acid practice was made. On several occasions when the silica had been exceptionally low the tuyères became hard to punch, and on turning the converter down to skim, it was seen that the adjacent brick were coated with some foreign material. Little significance was at first attached to this until in the latter part of January, 1913, it was noticed that this coating had spread and nearly all the brick were covered. Examination showed it to be very high in iron and magnetic, and it at once became evident that under certain conditions magnetic iron oxide was being formed in the converter and deposited



Measurements taken at intervals during life of lining to first patching. Matte average = 43.9 per cent. of copper.

A. After making 4,000,000 lb. of copper.
B. After making 6,000,000 lb. of copper.
C. After making 6,000,000 lb. of copper.
D. After making 10,000,000 lb. of copper.
E. After making 12,000,000 lb. of copper.
F. After making 14,501,962 lb. of copper.

Brick was kept coated with Fe_3O_4. Measurements A and B were taken with this coating in place. For all subsequent measurements it was, as far as possible, removed. Brick finally gave out at back about 20 in. above tuyères. Shell in operation from January 6 to June 27, 1913.

FIG. 2.—DIAGRAM SHOWING WEAR ON BRICK AT TUYÈRE LINE IN A 12-FT. GREAT FALLS TYPE BASIC-LINED CONVERTER.

on the brick. The thought immediately suggested itself that if this coating could be controlled at will it would, owing to its melting point being considerably higher than matte (about 1,525° C.), remain on the brick and greatly prolong the life of the lining by protecting it from all chemical and abrasive action. To attain this end the logical method was to blow matte without any silica present, thus oxidizing the iron to Fe_3O_4, when it was hoped that by later lowering the temperature of the charge by the addition of cold material, the magnetite would be precipitated and cover the interior of the converter. Experiments along these lines proved successful, and by the middle of February the shell could be coated at will. It was quite another matter, however, to keep the coating in place. Hard driving of the converter when the matte supply was large made proper running impossible, and patches of brick periodically became bare and the lining suffered in consequence. Finally after about four months' operating under these conditions, thorough control of the process was attained and bare brick in the interior of the converter became a thing of the past. The lining, however, in the meantime, had received such hard usage that shell No. 1 was taken off the stand to patch the brick and was replaced by shell No. 2.

This second shell, blown in on June 27, 1913, has benefited greatly by knowledge gained in operating the first and it has been possible to keep the magnetite coating intact from the time the shell was first started. After having made over 20,000,000 lb. of copper[1] in less than eight months, much of which was produced under conditions calculated to give the greatest wear to the lining, the brick shows but little sign of the strenuous campaign through which it has passed and the converter appears to be good for many million pounds more before a patching is necessary.

An increased tonnage to the furnaces has thrown more matte to the converter and it has been necessary to drive this shell much harder than the first. Over 50 per cent. of the time it is averaging better than 21 hr. actual blowing time out of the 24, and one charge follows another in rapid succession. Seventy tons of copper have been turned out in 24 hr., and for the period between Jan. 22 and Feb. 20, 1914, the converter averaged 57 tons of copper per day. Such work as this has been made possible only by keeping the brick covered with the magnetite coating and thereby protecting the lining from wear that would otherwise speedily ruin it.

To apply the magnetite covering, no hard and fast rule can be laid down, as the process must necessarily be varied in minor details to suit existing conditions. The following method is one frequently used at this plant: Into the new converter, previously heated by coal and coke to

[1] This shell has now been running steadily for more than a year and has turned out over 34,000,000 lb. of copper. It has not been necessary to patch or in any way repair the original lining.

expand the brick lining, a charge of about 12 tons of molten matte is poured and blown for about ½ hr. without any silica. By this time, most of the iron should be oxidized to Fe_3O_4, but, owing to the high temperature attained, the magnetite will be quite fluid and not apt to adhere to the brick. The addition of about 4 tons of cold hood cleanings or other material low in sulphur will lower the temperature sufficiently to cause the magnetic oxide of iron to become pasty and deposit on the lining. After this addition, blowing may be continued for a short time to spread the Fe_3O_4 evenly over the interior of the shell. More matte and just enough silica to flux the iron therein contained are added and the charge finished in the usual manner, care being taken to keep the temperature not higher than 1,200° C. This may be done by the addition of cold material when the charge begins to show excessive heat. Care must be taken, however, that the coating *is* Fe_3O_4, and not cold matte and slag, as a proper foundation on which to build is essential. Should the converter be allowed to get too cool these materials will coat the brick in places, and any Fe_3O_4 that may be applied subsequently will disappear as soon as the temperature rises above the melting point of these more fusible substances. This is a frequent source of failure and the cause of much uneven coating. By repeating the above treatment with each successive blow, a protective covering of any desired thickness can be applied. One 3 or 4 in. in depth should answer all purposes. When too thick, it makes tuyère punching very difficult. Once in place, the permanency of the coating is simply a matter of temperature control. In fact, it is impossible to prevent Fe_3O_4 from depositing on the interior of the converter when the practice is to make a very basic slag and the temperature is kept down, for a charge will occasionally be blown with insufficient silica and Fe_3O_4 formed and deposited on the brick. To cool the charge, an iron ore carrying 53 per cent. Fe, 10 per cent. SiO_2, and 2 per cent. Cu is frequently used, though cold slag, matte, hood or pit cleanings or scrap copper is equally satisfactory. Cold matte in quantity is not to be recommended unless one has the process well in hand, as the heat of oxidation is later apt to defeat the purpose for which it was added.

An average analysis of the converter slag made during the last eight months is: SiO_2, 22.7; Fe, 67.6; CaO, 1.0; Al_2O_3, 2.9 per cent. The matte averaged 43.5 per cent. Cu for the same period. As a matter of interest it may be mentioned that fluid slags have been made in coating the converter that gave the following analyses:

SiO_2 Per cent.	FeO Per cent.	CaO Per cent.	Al_2O_3 Per cent.	Cu Per cent.
4.2	83.2	0.2	0.8	6.0
10.0	81.1	0.3	0.9	4.2
11.0	79.2	0.3	1.1	4.1

Shell No. 2. First Campaign, June 27, 1913, to June 30, 1914

Commenced blowing	June 27, 1913
Taken off stand for patching	Still in use
Total blowing time	6,422 hr. 55 min.
Total number of blows made	1,499
Average time of blow	4 hr. 17 min.
Tons matte charged	39,080
Average copper content of matte	43.7 per cent.
Average time blowing one ton copper	24.1 min.
Total copper made	15,998 tons
Average copper per converter hour	2.5 tons
Average air used per minute	7,550 cu. ft.
Average blast pressure	12.6 lb.
Ore fed	8,486 tons

Average Slag Analysis	Per Cent.	Average Matte Analysis	Per Cent.
Cu	2.2	Cu	43.74
SiO_2	22.4	Fe	29.2
Fe	52.9	S	22.3
CaO	1.0	Ins	0.5
Al_2O_3	3.1		

Of the individual months' performances, May is most noteworthy, No. 2 shell having turned out 1,638 tons fine bullion from a 44.5 per cent. matte in a blowing period of 605 hr. 30 min., or a ton in 22.18 min. The blowing time in the above records is from the time when air is first received at the converter at commencement of the charge to the time the engine stops at the end of the charge. From 12 to 15 per cent. of this time is lost during skimming, charging, etc., and the air wasted during this time represents from 5 to 6 per cent. of the total air delivered by the compressor. From this it will be seen that the actual *oxidation period* of the charge is less than would appear from first glance at the above data, and the *true* oxygen efficiency of the converter is, therefore, somewhat greater than appears to be the case.

NOTE.—The approximate production of *fine copper* up to July 31, 1914, for shell No. 2 is 17,457 tons (or 54.6 tons.

DISCUSSION

E. P. MATHEWSON, Anaconda, Mont.—I would like to add a few words of praise on Mr. Howard's work at the Old Dominion. Mr. Howard has given this particular converter shell his personal attention, and the results are shown in the figures given. Similar converter shells, a little different in shape and of about the same size, have been tried in many plants throughout the United States. The results have been

highly satisfactory, but there is no shell of the size of the one described by Mr. Howard that has given anything like the results that Mr. Howard's shell has given. I know of larger shells that have given very good service, and are still in operation, and require very little repair. The main thing about the whole process of using the basic lining in copper converting is to keep the temperature within reasonable bounds, particularly not to get the furnace too hot. If it is overheated a little bit, for say half an hour, an entire lining may be ruined. It is very easy to keep the temperature regulated if the mass is large. If you have a large mass in your converter you can easily prevent it from getting overheated, and that is the big argument in favor of the extra large converters. Of course, there is a limit to what a plant can turn out in the way of matte. Very few plants can supply matte sufficient to keep a converter of 20 ft. in diameter in operation constantly, but the 20-ft. converter is very much superior to the 12-ft. converter in easy handling and absolute control of the temperature. I think Mr. Howard is to be congratulated on the excellent showing he has made.

The Ajo Copper-Mining District

BY IRA B. JORALEMON, WARREN, ARIZ.

(Salt Lake Meeting, August, 1914)

THE Ajo copper district is in the heart of the Arizona desert, near the western boundary of Pima county. Gila Bend, the nearest railroad point, is 43 miles north of the camp, and the little Mexican border town of Sonoita is 30 miles south. Between Gila Bend and Sonoita, Ajo is the only settlement save for one or two small cattle ranches and the uncertain villages of nomadic Papago families.

The Little Ajo mountains rise a few hundred feet above wide desert valleys, beyond which are lava *mesas*. The camp itself lies at an elevation of about 1,900 ft. above sea level, in a little basin on the east side of the range, separated from the open desert by low hills. In the center of the basin the brilliantly iron and copper stained rocks of Copper mountain rise 150 ft. above the village. Many varieties of cactus and low desert bushes cover the hills, while in the large valleys there are clusters of mesquite, paloverde, and ironwood trees which furnish limited amounts of fairly good fuel. A scanty supply of water is obtained from wells in the desert and from prospect pits in the camp. The nearest abundant source of water is the Gila river, 50 miles north of Ajo. (See Fig. 1.)

HISTORY OF THE AJO DISTRICT

The Ajo, next to Santa Rita, New Mexico, is claimed to have been the first copper district in the Southwest worked by Americans. In the '60's or even earlier high-grade native copper and cuprite ore was mined from shallow surface workings and hauled by bull team 400 miles across the desert to San Diego. From there it was carried in sailing ships to Swansea, Wales. Later, the ore was hauled to Yuma, floated down the Colorado river to the Gulf of California, and shipped to Swansea. Bat-infested old workings and the massive axles of the bull carts are shown to prove the truth of the stories.

From this early period until the beginning of the present century, the Ajo district was worked only in a casual and intermittent way. With the increased interest in copper mining during the past decade, the brilliant

surface showing at Ajo furnished an ideal basis for stock companies. A succession of reorganizations and new stock issues resulted in little underground development, and the greatest depth reached in the two or three years of activity was hardly more than 100 ft. Rich bornite and chalcocite ore was shipped from several small veins, and stamp mills with concentrating tables were installed to treat lower-grade ores. But the long dry haul to the railroad made the shipment even of rich ore and con-

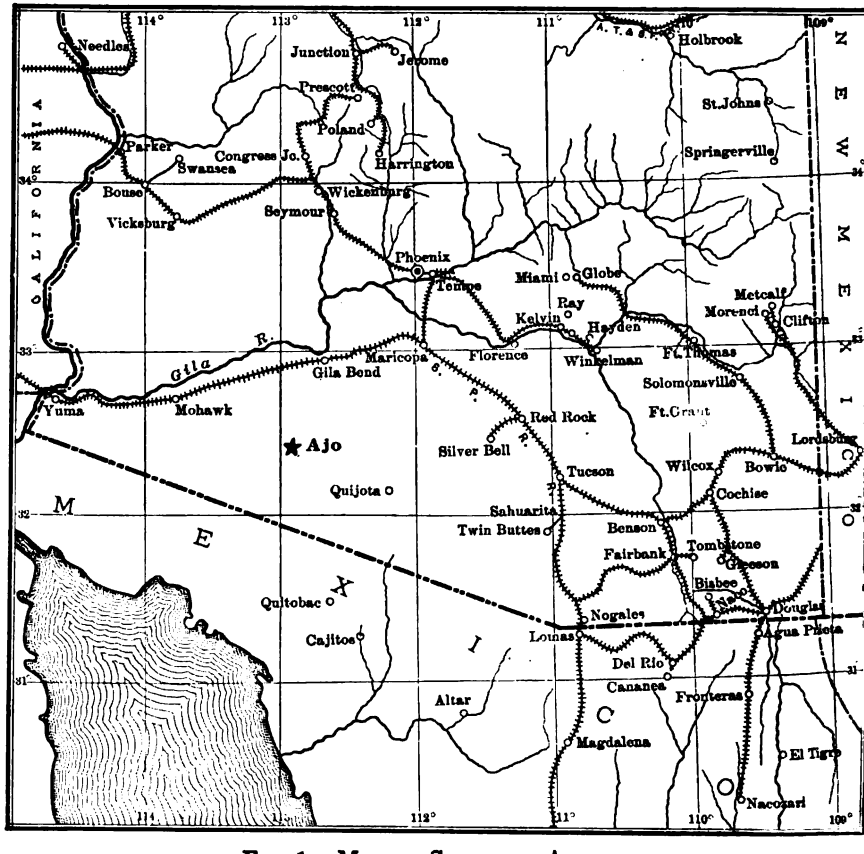


FIG. 1.—MAP OF SOUTHERN ARIZONA.

centrates hardly profitable. In an effort to avoid the cost of freighting ore, one of the companies fell a prey to the promoters of patent processes. The McGann vacuum smelter and a complicated hydrofluoric acid leaching plant still stand as monuments to the hopes of disappointed stockholders. The end of this period in the life of the district came with the panic of 1907, when the final products of the reorganizations—the New Cornelia Copper Co. and the Rendall Ore Reduction Co.—were forced to close down.

The Ajo camp enjoyed a second short-lived boom in the winter of 1909–1910. The Lewisohn interests secured an option on the New Cornelia Copper Co., and Seeley W. Mudd and associates optioned the Rendall Ore Reduction Co. Diamond drilling and underground work were started on the New Cornelia property, and churn drilling and underground work on the Rendall. Many engineers came to examine or buy smaller properties, and claims were located for miles in all directions. Since the drilling did not give satisfactory results, both options were soon given up. The camp returned to a moribund condition, and until the fall of 1911 no work was carried on save for a little leasing on small high-grade surface veins.

Nearly all of this early work was in the lower ground surrounding the hard silicified outcrops of Copper mountain. A few shallow holes on Copper mountain had developed low-grade, very siliceous malachite ore, and three or four deeper holes had penetrated below the carbonate zone to a disseminated chalcopyrite and bornite ore assaying from 2 to 4 per cent. copper. Both the carbonate and the sulphide ores were so unusual in character that it was doubtful if they could be treated with profit by processes in common use even if large orebodies existed.

In the fall of 1911 the Calumet & Arizona Mining Co., John C. Greenway, General Manager, took an option on all the available stock of the New Cornelia Copper Co., and started diamond drilling to prove more thoroughly what lay beneath the highly stained outcrops of Copper mountain. As work progressed favorably, test-pitting was started to increase the speed of development and to check the results of drilling. Later, drifting was done in the sulphide zone to prove the continuity of ore between drill holes. This work has shown that the silicified iron and copper stained hills are the outcrop of a great low-grade copper orebody, covering an area of about 55 acres and reaching a maximum depth of over 600 ft. below the surface.

After this development was begun, options were taken on the property of the Rendall Ore Reduction Co. by James Phillips, Utley Wedge and others under the name of the Ajo Copper Co., and on the Childs group of claims, between the New Cornelia and the Rendall properties, by representatives of the United States Smelting, Refining & Mining Co. The latter option was given up after a few churn-drill holes had been sunk. The Phillips interests still hold the old Rendall property, but are not at present doing any development.

Geology

Except for a local conglomerate, all the rocks in the Ajo district are igneous, and there is nothing to indicate the geologic ages. The earliest formation exposed is a series of rhyolite lava, breccia, and tuff beds.

These vary from white hard rhyolite to reddish, soft, coarse volcanic tuff, with the hard rhyolite beds usually near the bottom of the series. Next in age is an intrusion of monzonite porphyry, which cuts and uplifts the rhyolite. In the center of the district, the porphyry is a coarse quartz monzonite, with large orthoclase, plagioclase, and biotite crystals, and small variable amounts of quartz. The ground mass between crystals is generally white feldspar. This variety of the porphyry is distinguished by the pinkish orthoclase crystals. Further north, the porphyry grades to a granitic type, with less of the soda-lime feldspar, and much white orthoclase, biotite, and quartz. In the eastern part of the intrusion, the porphyry is finer grained and darker, without the pink feldspars. All through the porphyry mass there are local variations ranging from a fine-grained dark diorite to nearly white granite porphyry. Probably several minor intrusions closely followed the principal one, though the later dikes can seldom be distinguished either on the surface or in diamond-drill cores.

After the monzonite came a few dikes of diorite or diabase porphyry, usually dark gray in color, with square crystals of white feldspar in a very fine-grained ground mass. These are probably allied with the great mass of Tertiary andesite and basalt lava flows which cover hundreds of square miles of the surrounding desert region. This lava surrounds the Little Ajo mountains on all sides, beyond desert valleys, and small outcrops of it are left above the desert wash immediately northeast of the district. Probably at one time it covered the whole country. In a drilled well in the valley 6 miles northeast of Ajo, the thickness of andesite and basalt lava flows has been proved to be over 1,200 ft.

The most recent rock found at Ajo is a coarse conglomerate which lies between the hills and the valley to the east and south. The fragments in this conglomerate are rhyolite and monzonite, and the cementing material is limonite and silica, with traces of malachite. The age relation between the conglomerate and the Tertiary lava flows is not certain. Evidently the conglomerate is a local result of rapid erosion of the mineralized rhyolite and monzonite of Copper mountain, formed by the cementing by iron and silica bearing surface waters of rocks washed down the steep mountain side.

The only great alteration of the rocks, other than that accompanying the mineralization, is along rhyolite-monzonite contacts. Here the monzonite becomes much finer grained, and certain rhyolite beds have been recrystallized into speckled, fine-grained, gray crystalline rocks. The recrystallized rhyolite is sometimes hard to distinguish from fine-grained varieties of monzonite, especially where inclusions of rhyolite in monzonite are completely recrystallized. In the eastern part of the basin, the recrystallization of occasional beds of rhyolite strongly suggests dikes of fine-grained crystalline rock following rhyolite bedding.

Structure

The most important feature of the geologic structure is the large laccolith or batholith of monzonite porphyry, which uplifted the older rhyolite beds to form a dome. The crest of the dome was eroded away at a fairly recent time. This porphyry mass is 8 or 10 miles long by from 1 to 4 miles wide, with the long axis extending N. 20° W. The northern part of the intrusion forms high rocky hills north of the Ajo basin. The southern end forms Copper mountain, in the center of the basin. The copper deposit occurs in this southern part of the monzonite, where the axis of the laccolith is plunging to the southeast beneath the rhyolite. On top of Copper mountain, the small remnants of rhyolite lie fairly flat. On the west flank of the intrusion the dip of the rhyolite is steeply to the southwest, while the beds on the east flank dip to the southeast at an angle of about 20°. The pitch to the south of the domed rhyolite beds is fairly steep. All of the contacts are irregular.

Copper mountain and the Ajo basin surrounding it are thus formed by the monzonite and by the contact beds of rhyolite. East and west of the basin are hills of rhyolite lava, breccia, and tuff, with flat-lying beds of andesite lava and basalt in the distance, usually beyond gravel-filled valleys. South of the basin a large body of the red-brown conglomerate stretches away to the dark andesite and basalt mass of Black mountain.

There are many inclusions of rhyolite within the monzonite, and dikes or irregular intrusions of monzonite break up through the rhyolite, especially east of Copper mountain. Diorite or diabase dikes cut both monzonite and rhyolite, but apparently have had no important effect on structure or mineralization.

Both the monzonite and the rhyolite near it are thoroughly shattered by fractures which run in all directions. While some of these fractures are accompanied by considerable gouge, and may be faults of some importance, most of them can be traced for only a short distance, and are probably contraction fissures. In any portion of the porphyry there is generally a well-marked direction of strongest fracturing, but in the porphyry mass as a whole no such generalization can be made.

The somewhat idealized east-west sections (Fig. 2) show the most important points in the geologic structure of the Ajo basin.

Mineralization

The Disseminated Orebody

The mineralization in the Ajo district has formed a low-grade disseminated copper deposit with higher-grade veins in the monzonite, and narrow rich veins in the adjoining rhyolite. In the monzonite, the dis-

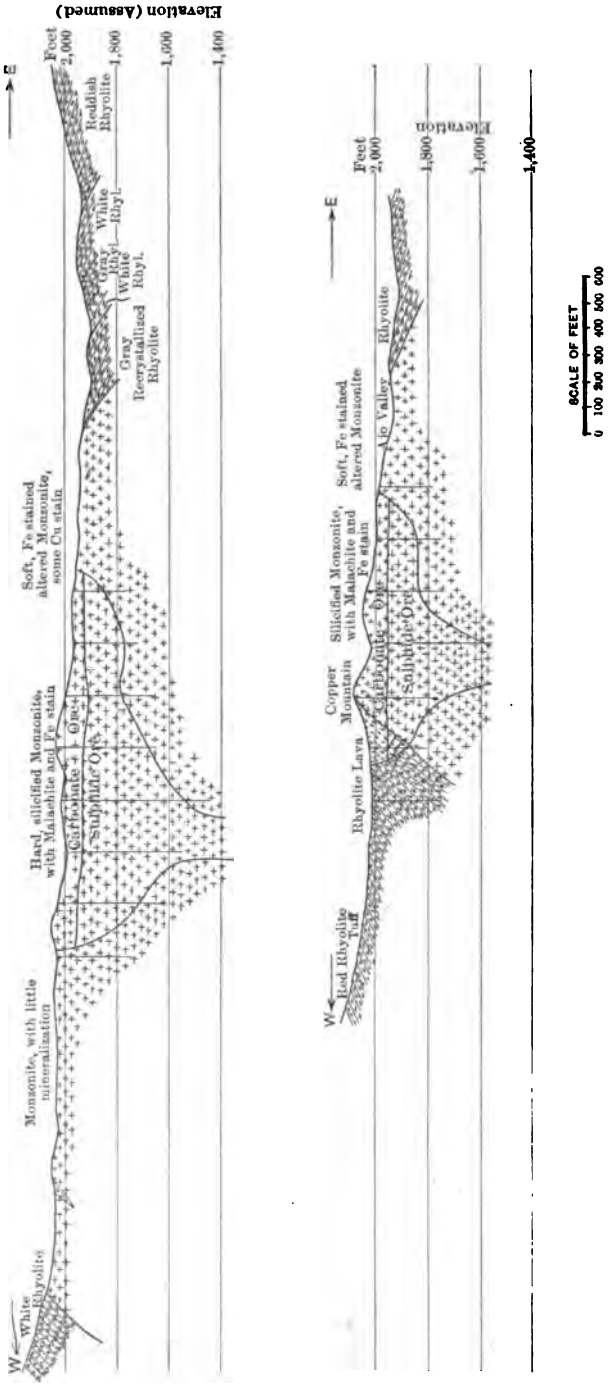


FIG. 2.—VERTICAL SECTIONS THROUGH NEW CORNELIA OREBODY.

seminated deposit has a roughly pear-shaped outline, with the neck of the pear to the south. The outline of the orebody agrees almost exactly with that of Copper mountain and of the silicified, iron-stained hills north of the mountain. The area covered by the orebody is about 55 acres. The depth of the ore varies greatly. Around the outskirts of the orebody it often extends less than 50 ft. below the surface, while in the center drill holes from 400 to 600 ft. deep have not found the bottom of the ore. In vertical section the deposit has the form of a gigantic elongated mushroom, the stem of which reaches below any point yet reached in drilling. The accompanying vertical sections give a fair idea of the form of the orebody.

Unlike Bingham, Ely, Santa Rita, and other low-grade camps, in which the copper is in the form of chalcocite, the disseminated ore in the Ajo district is a mineralization of the porphyry with chalcopyrite and bornite. Within the orebody, the monzonite is thoroughly shattered by a network of fractures. While the larger breaks often run about N. 20° W., parallel with the axis of the intrusion, other fractures have all possible directions. Along many of the fractures there are quartz veins from ¼ in. to 1 ft. or more wide. Between these veins the monzonite is more or less silicified, being often completely replaced by quartz. With the quartz, chalcopyrite and bornite were introduced into the mass, both in narrow seams or films along fractures and as very small flakes, disseminated through the porphyry. In most cases both the disseminated sulphides and those in seams or veinlets are required to bring the ore up to commercial grade. Within the main portion of the orebody pyrite is entirely absent or is present only in very small quantities. Chalcocite is rarely found except in very thin films just below the water level. There is a small and variable amount of magnetite disseminated through the ore. This is apparently earlier than the copper minerals, and may be an accessory original constituent of the rock.

Along some of the larger fractures the seams of bornite and chalcopyrite widen to veins an inch or more across. Where several of these veins lie close together and parallel, there are bands of ore from 10 to over 100 ft. wide, assaying from 3 to 5 per cent. copper. The grade of the ore in general is exceedingly variable, changing abruptly from slightly mineralized porphyry containing less than 0.5 per cent. copper to ore assaying over 3 per cent. As would be expected, the richer ore is in the more thoroughly fractured portions of the porphyry. The divisions of rich and lean material are often in more or less parallel bands, following in any part of the orebody the general direction of fracturing at that point. Since the bands are so narrow and irregular that the richer ore could not be sorted out without excessive expense, the whole mass has been considered as a solid orebody, the grade of which is lowered by the lean material included in it.

Sulphide Ore Primary

While in different parts of the orebody, and often within a few feet, the relative proportions of chalcopyrite and bornite vary greatly, on the whole the two minerals continue in approximately the same proportions in the deepest ore developed as in that a few feet below the oxidized zone. In the deepest ore there is no indication of any action by circulating surface waters. The general occurrence of the chalcopyrite and bornite seems to indicate that both minerals were formed during the original mineralization, and were not at all due to enrichment by solutions descending from the surface. L. C. Graton has examined in polished section some of the richer bornite ore from the New Cornelia orebody, and concludes that the bornite and chalcopyrite are both primary.[1] The variations in grade are due to differences in the intensity of the original mineralization, not to any effect of surface enrichment. In this fact lies the chief difference between the New Cornelia orebody and the disseminated deposits now being mined in other districts.

Boundaries of Orebody Commercial

The boundaries of the orebody at Ajo are generally commercial. On the southwest side, the ore in the monzonite extends to the contact with the overlying rhyolite. In places the rhyolite beds for some distance from the contact are mineralized strongly enough to form an ore of commercial grade. But more often the mineralization in the rhyolite is confined to narrow films of chalcopyrite and bornite on joint planes, and the grade of the material is under 1 per cent. East of the orebody, the change from ore to lean material is caused not so much by a decrease in the amount of mineralization as by a change from chalcopyrite and bornite to pyrite. There is less silicification in the pyritic porphyry than in the ore, and the dioritic type of porphyry is more common. West and northwest of the ore, the change from commercial ore to material too lean to be considered ore is caused generally by a decrease in the amount of chalcopyrite, bornite, and quartz, with no great change in the character of the mineralization. West of the ore, the porphyry changes from the typical monzonite to a dioritic phase, while to the north there is a gradation to a more granitic porphyry.

Below the outer portions of the orebody, where the bottom of the ore has been developed, drill cores show that the ore gives place to less fractured, less highly silicified monzonite, containing very little chalcopyrite and bornite. Occasionally a sudden change to a more acid or more basic variation of the porphyry is accompanied by a drop from ore to lean

[1] L. C. Graton: The Sulphide Ores of Copper, *Trans.*, xlv (1913), Figs. 18 and 26, pp. 75 and 79; also private communication.

material. The association of the ore with the normal, coarse-grained monzonite is very noticeable. Since the variations in the rock seem too slight to have had any selective influence on the precipitation of metallic minerals, it is probable that the formation of the coarse monzonite type of porphyry and the deposition of the ore were both dependent on the conditions existing at the top of the intrusive mass, below the uplifted rhyolite.

Surface Alteration

The result of alteration by surface waters at Ajo is entirely unlike that in other low-grade copper districts. Instead of having an oxidized capping from which nearly all the copper content has been leached, the sulphide ore is capped by crushed, silicified monzonite, shot through with seams and stains of malachite, limonite, hematite, and a little chrysocolla. The rock is still hard, though the feldspars are kaolinized to a greater or less extent. Often kaolinized feldspar crystals are stained bright green by malachite. The actual surface of the hills is tinted by iron oxides a deep red brown, with brilliant copper staining in protected places under the cliffs and wherever the surface rock has been shot away. Over an area of 30 or 40 acres it is hard to find even a small piece of rock which on breaking does not show copper stains and seams of malachite. While there is a little chrysocolla in veinlets, and occasionally cuprite, chalcocite, and bornite in richer veins, it is safe to say that over 85 per cent. of the copper in the oxidized zone is in the form of malachite. Now and then disseminated chalcopyrite and bornite are left unaltered in hard ore between fractures. But such remnants of sulphides are unimportant compared with the great mass of carbonate ore.

As in the case of sulphide ore, the assay value of the carbonate ore varies greatly. Rich and lean bands alternate. Sometimes, along large fractures, the rock is softened and leached, with less copper than in the adjoining harder rock. But more commonly the variation in values is not accompanied by any change in the degree of alteration of the rock. On the average, the copper content of the oxidized zone is constant from the surface to the bottom of the zone, and is almost exactly the same as that of the underlying sulphide ore.

The oxidized ore continues down to an almost horizontal plane, about 20 ft. below the deepest *arroyos* and 150 ft. below the highest hills. The remarkable regularity of the dividing line between sulphide and carbonate ore is shown by the vertical section. This line, or rather plane of demarcation, agrees almost exactly with the present ground-water level. The transition from carbonate to sulphide ore is very abrupt. Generally drill cores show less than 5 ft. of partly oxidized ore, the classification of which is doubtful. Occasionally, where leaching is unusually great along large fractures, bands of carbonate and sulphide ore alternate for

15 or 20 ft. vertically. But on the whole, the amount of sulphide ore above the plane, or of carbonate ore below it, is very small. There is also no appreciable enrichment at the top of the sulphide zone, with a few local exceptions. This sudden change on a horizontal plane from carbonate to sulphide ore will greatly simplify both the mining and the treatment of the New Cornelia ore.

East of the orebody, where the disseminated copper sulphides give place to pyrite, the oxidation extends to practically the same horizontal plane at water level. The character of the oxidized zone, however, is entirely different. Instead of being hard and siliceous, the rock is soft, with much kaolin and soft yellow limonite. Copper staining is comparatively slight. Instead of standing up in bold brown and green cliffs, the outcrops are reddish and form valleys or low hills. The oxidized zone contains much more limonite and alumina, but less silica and very little copper. At the top of the lean pyritic material there is a 5 or 10 ft. zone of enriched chalcocite ore assaying from 1.5 to 3 per cent. copper. In short, the conditions in this part of the Ajo basin are almost exactly similar to those in the great disseminated chalcocite districts, except that the primary mineralization was too lean in copper, or the leached zone was too shallow, to form an enriched orebody of commercial size.

Veins in Rhyolite

It was not the low-grade orebody, but the rich veins in the surrounding rhyolite, which first led to the exploitation of the Ajo district. One strong vein outcrops on the Quien Sabe claim of the New Cornelia Copper Co., on the lower slopes of Arkansas mountain, and four or five veins have been partly developed on the Ajo Copper Co. property. Rich malachite and cuprite ore from 6 in. to 3 ft. wide outcrops in these veins, which follow steep fractures in hard, slightly iron and copper stained rhyolite. High-grade cuprite and copper-glance ore, with a little native copper, was encountered a few feet below the surface. At a depth of about 50 ft., the glance begins to give place to bornite. Usually the center of the vein is very rich bornite and chalcocite ore from 1 in. to 4 or 5 ft. wide, and on both sides of this high-grade streak the shattered rock contains stringers of bornite and chalcocite, which make it a good concentrating ore. In the early operations in the district, considerable stoping was done in several of the veins to a maximum depth of over 100 ft. The stopes are seldom more than 6 ft. wide, and the large dumps show that much of the material taken from stopes was too low grade to treat with profit. In the Ajo Copper Co. property, one rich bornite vein from 1 to 3 ft. wide was developed for nearly 300 ft. down the dip. The high-grade stringer continues to the bottom, but the mineralization of the walls appears to decrease in depth.

Most of the veins are associated for at least part of their length with dikes of monzonite, which tend to widen with depth. The monzonite adjacent to the veins is often mineralized enough to form a 2 or 3 per cent. ore, with values in malachite near the surface, and in chalcocite and bornite below a depth of 50 to 75 ft. In one or two places, lenses of this disseminated ore in monzonite dikes reach a width of over 50 ft., and promise to yield a considerable tonnage. They have as yet not been developed enough to prove how deep the ore will continue, and whether or not they are of great importance.

Genesis of Ore and Geologic History

The genesis and geologic history of the Ajo ore seem unusually easy to trace. After the monzonite intrusion had uplifted the rhyolite, the slow cooling of the porphyry was accompanied by considerable contraction. This resulted in a thorough jointing and fissuring of the monzonite, especially near the rhyolite contact, and in a less complete fracturing of the rhyolite itself. Near the center of the intrusion, some of the fissures continued to great depth. Probably soon after the solidification of the outer layer of porphyry, hot mineral-bearing solutions rose along these deep fractures. The solutions were heavily charged with iron, sulphur, silica, and later copper. The iron and sulphur were carried through the joints and fissures in the rock far from the source of the mineralization, and caused a general, though scanty, pyritization of the monzonite mass. This pyritization was heaviest in the eastern part of the intrusion. Later, as the proportion of silica and copper in the solutions increased, these solutions rose through the larger fractures in the monzonite until they encountered the impervious, less thoroughly fractured overlying beds of rhyolite. These dome-shaped beds acted as a dam, stopping the upward flow of the mineralizing solutions and causing them to spread out through the jointed monzonite on both sides of the large, deep-seated fractures. Here they remained imprisoned until they gave up their mineral content, depositing veins of quartz, chalcopyrite, and bornite along the fissures and joint planes, and partly replacing the rock itself with the same minerals. The mineralization was greatest near the large central fractures through which the solutions had risen, and in the more thoroughly jointed portions of the monzonite near rhyolite contacts. In this manner was formed the mushroom-shaped disseminated orebody, grading on the sides and bottom to rock less thoroughly mineralized, or mineralized with iron instead of copper sulphides.

Some of the larger fractures, usually accompanied by monzonite dikes, extended for a considerable distance up into the rhyolite. Along these fractures the rich bornite veins in rhyolite were deposited, sometimes extending a long distance from the large disseminated body. The por-

phyry dikes accompanying the veins were more or less mineralized, and small quantities of chalcopyrite, bornite, and pyrite were deposited in the rhyolite walls of the veins.

Probably soon after the end of the period of mineralization, before the rhyolite capping of the ore had been eroded away, the whole country was covered by a great thickness of andesite lava and basalt. This covering protected the ore from surface alteration until the present geologic period.

Erosion in early Quaternary times must have been very rapid. Deep canyons were cut in the recent lavas. The covering was stripped off from the Ajo monzonite, and much of the orebody itself was rapidly washed away. The erosion was so rapid that oxidation did not keep pace with it. The copper in the ore that was eroded away was either left in the rock fragments washed down the hillsides or was carried off by the surface drainage. The detritus from the eroded portion of the orebody filled the canyons and valleys that had been cut in the lava east and south of Ajo. Part of the detritus was cemented into a hard ferruginous conglomerate by surface water which had flowed over the pyritic porphyry east of the ore and had taken into solution some of the iron of the oxidized pyrite. Boulders of oxidized ore remain scattered through the conglomerate and through the valley wash or gravel for several miles from Ajo.

Following this period of rapid erosion, very stable conditions have lasted up to the present time. The water level has remained nearly stationary, at a rather shallow depth, long enough to allow thorough oxidation of the rock down to this level. Where the mineralization was chiefly with pyrite, much sulphuric acid was formed by the oxidation of the pyrite to limonite. This acid thoroughly softened and kaolinized the feldspars of the monzonite, and helped take into solution the copper from the traces of chalcopyrite which accompanied the pyrite. This copper was carried down by the descending surface waters and was redeposited at water level to form the thin layer of chalcocite ore.

In the siliceous chalcopyrite and bornite orebody, it seems that little sulphuric acid was formed during the oxidation. The descending surface water, probably always small in amount in that arid climate, contained an excess of lime and of carbonic acid. The result of this condition was that as soon as the copper sulphides were oxidized, the copper was precipitated almost in place in the form of malachite. The iron of the chalcopyrite and bornite remained in place in the form of limonite and hematite. The silicified monzonite was very slightly altered, because of the small quantities of sulphuric acid, and remained hard and resistant to weathering, forming steep hills. The little sulphuric acid which was formed probably reacted at once with the calcium carbonate in the oxidizing waters to form calcium sulphate. Thin seams of the resulting gypsum occur here and there in the carbonate ore, and there is much gypsum in the gravel of the valleys.

The result of this cycle of changes was a large body of low-grade primary bornite and chalcopyrite ore, oxidized above a horizontal plane at the present water level to a malachite ore of exactly the same grade as the sulphide. The ore was deposited from rising deep-seated solutions. There is no overburden of lean or barren rock, since the oxidation was accompanied by practically no leaching or transportation of the copper from the primary ore. The ore rises above the surrounding barren ground, and continues without any considerable increase or decrease in values from the actual surface to the bottom of the orebody.

Development of the Low-Grade Orebody

The Calumet & Arizona Mining Co. developed the disseminated orebody on the New Cornelia property by diamond-drill holes and by test pits. The probable ore-bearing ground was co-ordinated with east-west and north-south lines at 200-ft. intervals, and drill holes were sunk at the intersections of co-ordinate lines. The drilling was done under contract by the E. J. Longyear Co., of Minneapolis, Minn. Sampling was under the direction of representatives of the Calumet & Arizona Mining Co. Drill holes were sampled in 5-ft. sections. All of the flow of sludge was caught in barrels and settled. When the end of the 5-ft. section was reached the water in the barrels was decanted off and the sludge sample was dried and quartered down to 3 or 4 lb. The rods were pulled at least every 5 ft., and core samples were taken at even 5-ft. intervals where possible. Both core and sludge samples were sent to the assay office of the Calumet & Arizona Mining Co., in Bisbee, for analysis. Small portions of both core and sludge for every 5 ft. of drilling were kept for future reference in labeled tin boxes.

Owing to the thoroughly fractured, uneven nature of the rock, the recovery of core was low, and neither core nor sludge samples alone were satisfactory. To obtain an accurate assay value for the ore developed, the length of core for every 5-ft. advance was measured, and on the basis of this length of core the sludge and core assays were combined to give a final value which represented all the material removed from the hole during the 5-ft. advance. The E. J. Longyear Co. furnished a chart which greatly simplified the work of combining core and sludge samples.

After drilling had been in progress for about six months, test-pitting was started to check the results of drilling. The pits were about 4 by 6 ft. in size, and were sunk with windlasses. Drilling was done by hand. Mexican and Papago Indian miners were employed for this work, usually under contract, and soon became so efficient that to a depth of 50 ft. the test pits were sunk more cheaply than diamond-drill holes. During the last year of development, test pits 50 ft. deep were sunk on co-ordinate points in advance of drilling, in order both to expedite and to lessen the cost of development.

In sinking test pits, every tenth bucket windlassed from the hole was taken for a sample. The large samples thus obtained for every round shot were crushed, quartered, and sent to Bisbee for analysis. After test-pitting was finished, the pits were resampled in 5-ft. sections by channeling. All four sides of the pits were channeled with even rectangular vertical grooves 6 in. wide by 3 in. deep, giving about a 500-lb. sample for every 5-ft. section. These samples were crushed, quartered, and assayed at Ajo, and pulp samples were reassayed in Bisbee. The channel samples averaged about 0.15 per cent. lower than the bucket samples and were taken as the final samples of the test pits.

When test pits were sunk below water level into sulphide ore, a small flow of water was encountered. This made sinking slow and expensive. Therefore only a few hundred feet of sinking was done in sulphide ore. To more thoroughly check the results of drilling in sulphide ore, and to prove that the ore was constant in grade between drill holes, drifts were run on co-ordinate lines from the bottoms of two of the deepest test pits. Raises were put up from these drifts to check drill holes. To give a fair sample of material cut by the drifts, the ore broken by every round of holes was hoisted separately and dumped on an iron plate. From here it was shoveled into cars or over the dump, and every tenth shovel was thrown into a bin for the sample. These large samples, each representing a carefully measured round, were taken to the mill, crushed, quartered, and sent to Bisbee for analysis.

Up to September, 1913, when development work was stopped, 84 diamond-drill holes had been sunk, varying in depth from 200 to 1,000 ft. The total footage of diamond drilling was 23,097 ft. Nineteen drill holes were stopped in ore.

In all, 77 test pits were sunk, with a total footage of 3,955 ft. Of this test-pitting, 3,606 ft. were in carbonate ore, and 349 ft. in sulphide ore; 1,059 ft. of test-pitting checked drill holes in carbonate ore, 175 ft. checked drill holes in sulphide ore, and 2,721 ft. were in advance of drilling. The total amount of drifting in sulphide ore was 1,513 ft. The combined sinking and raising on drill holes in sulphide ore amounted to 317 ft.

Results of Development

The sinking, drifting, and raising proved that the value of ore indicated by drilling was very accurate. The channel samples of test pits in carbonate ore averaged 0.005 per cent. lower than corresponding diamond-drill samples; the test pits and raises in sulphide ore averaged 0.05 per cent. lower than corresponding diamond-drill samples; and the drifts in sulphide ore averaged 0.26 per cent. higher than the assay value of blocks of ore through which the drifts were run, as indicated by drill holes at the corners of the blocks. In making the ore estimate, diamond-

drill samples were accepted wherever drilling was done, and channel samples of test pits were accepted where sinking was done in advance of drilling.

Ore Estimate

The estimate of ore developed on the property of the New Cornelia Copper Co. at Ajo by the Calumet & Arizona Mining Co. is as follows:

Ore estimate:	Tons	Copper, Per Cent.
Carbonate	11,954,400	1.54
Sulphide	28,303,600	1.50
Total	40,258,000	1.51
Available by steam shovel:		
Carbonate	11,954,400	1.54
Sulphide	20,526,800	1.54
Total	32,481,200	1.54
Not available by steam shovel:		
Sulphide	7,776,800	1.40
Rock which must be removed to make steam-shovel tonnage available:		
Rock in carbonate zone	708,400	0.65
Rock in sulphide zone	2,600,000	0.63
Total rock	3,308,400	0.63

The gold and silver content of this ore is very low, generally amounting to less than 15c. per ton.

In computing the ore tonnage, no ore was included which assayed less than 1 per cent. copper. The estimate of ore available for steam-shovel mining depends, of course, on the assumed maximum grade of tracks in approaches and pit, and on the amount of lean rock in the sides and bottom of the pit which it will pay to move in order to give access to deep ore. The proportion of the Ajo ore mined by steam shovel will probably be greater rather than less than that given in the above estimate.

The fact that the Ajo ore outcrops at the surface, with no barren overburden, not only will do away with the high initial stripping cost which generally attends steam-shovel mining, but also will prevent a loss of ore or a vitiation of grade through a mixture of ore with capping. This feature should make the mining grade and tonnage agree very closely with the estimate.

Treatment of Ores

While the question of the treatment of the New Cornelia ores has not as yet been fully settled, enough work has been done to prove that the ore can be treated at a good profit by processes in use in other districts. Experiments on ore treatment have been carried on in Douglas and elsewhere for about a year and a half. This work has been under the direction of John C. Greenway, General Manager, with the advice of Dr. L. D. Ricketts. The first important tests were made by Stuart Croasdale

on leaching carbonate ores. These tests showed that over 80 per cent. of the copper in carbonate ore crushed to ½ in. can be leached out by sulphuric acid in from 36 to 72 hr., and that this copper can be precipitated at a reasonable cost by scrap or granulated pig iron. Good percolation was secured in leaching tanks with an ore column over 10 ft. high. Owing to the very high insoluble content of the ore, the acid consumption was low enough so that regeneration of sulphuric acid or ferric sulphate from the solutions from which the copper has been precipitated is not necessary to the success of the process. When it had been shown that leaching with fresh sulphuric acid and precipitation on metallic iron would yield a fair profit on the carbonate ores, experiments were directed toward lessening the cost of leaching and precipitation by using processes in less common use. The making of sponge iron for a precipitant by metallizing the calcines obtained by roasting pyrite ores was successful only when tried on a very small scale. A process developed by F. L. Antisell, of the Raritan refinery, involving electrolytic precipitation with an insoluble anode made largely of coke, and regeneration of ferric sulphate, has been very successful on a small scale. A. D. Jamieson, who was associated with Mr. Antisell in these experiments, is now testing this process on a larger scale in Douglas. Thus far the work has been successful. Utley Wedge has met with encouraging results in a process involving a sulphatizing roast of mixed carbonate and sulphide ores, with subsequent leaching. It seems reasonably certain that one of these newer processes will make possible a great saving in cost of copper over simple leaching with sulphuric acid and precipitation on iron.

The Ajo sulphide ores can be concentrated by ordinary wet methods, but the thin flakes of bornite and chalcopyrite slime badly, and the recovery is low. The chalcopyrite and bornite ore is beautifully adapted to flotation, and tests made by the Minerals Separation Co. show a high recovery. Very fine crushing is necessary to allow the separation of the sulphides, and the hardness of the ore will make the crushing fairly expensive. It seems certain, however, that sulphide as well as carbonate ore can be mined and treated with a good recovery at great profit.

Future Plans of the New Cornelia Copper Co.

With a great tonnage of ore developed, and the general outline of the method of treatment in sight, the most important problem at Ajo is the development of a sufficient supply of water for the mill. To solve this question, two deep wells were drilled in the large valley northeast of Ajo. Water was found in both wells at depths of from 550 to 750 ft. A boiler plant and compressors were installed at one well, and the water was pumped from the drill hole by an air lift. The limit of capacity of the lift was about 175 gal. per minute. A pumping test at this rate lasting over two weeks did not exhaust the water from the hole. With this flow

from an 8-in. drill hole, it is reasonably certain that a shaft, with a little drifting, will furnish an ample supply of water. A shaft has been started for this purpose.

To determine the details of treatment, a large experimental plant is being started at Ajo. The units in this plant will be of practically the same size as those in the final plant. Large leaching tanks will be erected, and an electrolytic precipitation plant. Probably both the Antisell process and another process involving electrolytic precipitation, devised by Messrs. Pope and Hahn, of New York, will be tested in this experimental mill. Since it will be several years before mining of the carbonate ores proceeds far enough to expose sulphide ore, the development of a process for treating sulphide ore can be carried on more leisurely.

FIG. 3.—NEW CORNELIA OREBODY, AJO. LOOKING NORTH.

A railroad will be the next step in bringing the New Cornelia property to the productive stage. Preliminary surveys have been made from Ajo both to Gila, on the Southern Pacific Railroad, and to Tucson, the junction point of the Southern Pacific and the El Paso and Southwestern systems. The distance to Gila is about 44 miles, and to Tucson about 130 miles. Both lines are through gently sloping desert country, and railroad building should cost less than $20,000 per mile. It has not yet been determined which route the railroad will follow.

The orebody already developed at Ajo will supply a 4,000-ton mill for over 26 years, or a 6,000-ton mill for 18 years. The indications are that the life of the mine will be greatly lengthened by the development of a large tonnage of deep ore along the fracture zone in the center of the orebody. The vital points in the question of ore treatment have apparently been successfully solved. For the next quarter of a century the Ajo will be one of the great copper districts of the Southwest.

Leaching Experiments on the Ajo Ores

BY STUART CROASDALE, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

NOT long ago I was called upon to conduct some experiments on the treatment of ores from the New Cornelia copper mine, Ajo mountains, Arizona, for the Calumet & Arizona Copper Co. The problem was a very interesting one and contained some unusual features. Incidental with this problem was the utilization of a large amount of low-grade pyrite in the Bisbee mines of the company, provided this could be accomplished by any feasible method.

The New Cornelia mine is situated 45 miles south of Gila Bend, the nearest railroad point, on the Southern Pacific Railroad. It is about 160 miles, almost due west, from Tucson. This section is one of the most arid in the United States. The normal annual precipitation, as given by the United States Weather Bureau covering a period from 1870 to 1901, is 3 to 5 in. During June and July, 1913, the records kept by the company gave the following results. These are all I have available, but they show the enormous evaporation that can be expected, which may be a factor in leaching operations on a large scale.

	Temperature, Degrees F.		
	High	Low	Average
June	112	62	84.0
July	115	71	87.8

Evaporation	June	July
Concrete tank, inches	11.7	10.72
Iron tank, inches	14.8	13.07
Rainfall, inches	None	0.77
Cloudy days	None	10.00

Aside from several springs about 10 miles from the property, the nearest known water in quantity at the time these experiments were made was the Gila river at Gila Bend, 45 miles distant. Part of the development of the property of course included drilling for water in the valley, which was in progress at the conclusion of my experiments.

The ore deposit itself is unusual. It consists of an intrusive plug of monzonite or granite porphyry. Chalcopyrite, bornite, and magnetite are finely disseminated throughout the magma in sufficient quantities to make the greater part of the mass commercial ore to a certain depth.

Pyrite is also present, and when the proportion of pyrite has exceeded chalcopyrite in the oxidized zone the kaolinization of the porphyry has been greater, which has an important bearing on the ore treatment. Erosion has removed the surrounding country rock and left the plug sticking out at the base of the mountains as two or three cone-shaped buttes, without overburden of any description.

Oxidation has taken place *in situ*. Owing to the aridity of the country, there has been no leaching and no secondary enrichment. Instead of following the contour of the surface, as might be expected, the drill records (in the mine model) show oxidation to have taken place quite uniformly downward throughout the mass and the division between the oxidized and sulphide ores is almost a horizontal plane. The transition zone, or partly oxidized zone, between the oxidized and sulphide ores is likewise shown to be quite narrow—much narrower than would be expected.

The oxidized ores amount to about one-third of the total ore developed. The greater part of the copper exists as carbonate, although there is some silicate and some oxides, including cuprite. There is also a small percentage of sulphide as residual chalcopyrite and intermediate forms.

The oxidized iron exists in both ferrous and ferric condition, both soluble in acid.

While the chalcopyrite is quite generally disseminated throughout the magma, there seems to have been a tendency for it to segregate in some portions, thus producing a higher-grade ore. The average grade of ore used for experimental purposes was placed between 1.5 and 2.0 per cent. copper. What the ultimate average grade will be will depend upon the cost of treatment.

Methods of Treatment

From the foregoing it will be readily seen that the conditions of the problem demand simplicity of process and large scale of operations. Of course a railroad will be built to the mine, but whether the ore will be brought to water or water will be brought to the ore are yet factors to be considered and may have their bearing to a certain extent on the method of treatment adopted, when freight and fuel costs are estimated.

Methods of leaching with solutions of reducible salts, like ferric sulphate, ferric chloride, and cupric chloride, have some advantage in their solvent power on cuprite and chalcocite or other intermediate sulphides, but have little action on chalcopyrite. These salts are expensive and their value as commercial solvents depends upon their regeneration by some cheap method of oxidation after the dissolved copper has been precipitated. This difficulty has not yet been overcome and

the results from attempts to use these methods have been disappointing even in the hands of skilled metallurgists.

The same may be said of sulphurous acid and ammonia processes.

Two methods of treatment, however, are available, and both are entirely reliable and feasible under the conditions involved. One of these is the old Henderson process of roasting with salt, and the other is leaching the raw ore with sulphuric acid. Under the head of "Chloridizing Roast with Salt," I include the so-called sulphatizing roast of Mr. Wedge, in which he finds the addition of a small percentage of salt very beneficial; and also the modification used by Mr. Laist at Anaconda, in which he reduces the consumption of salt, as well as the losses by volatilization, by giving the ore a preliminary roast before adding the salt.[1]

A comparison of these two methods may be made briefly as follows:

By using the chloridizing roast, all ores on the property could be treated by the same process—oxidized, sulphide, and intermediate. This would mean the construction of only one plant. The average extraction of the copper would probably be higher than that obtained from acid leaching and the small amount of silver in the ore would be recovered to a large extent. By mixing the ores, little if any sulphur would have to be added as pyrite. The only chemicals required would be salt, to the extent of 5 to 10 per cent. of the ore treated, and possibly a small amount of pyritic ore. Both of these could be transported in ordinary freight cars. There would be less corrosive solutions to handle. There would be a minimum amount of iron and alumina passing into solution. There would be no appreciable absorption of copper by slimes.

On the other hand, this method would require dry crushing of the ore to 20 or 40 mesh, which would be troublesome if not prohibitive on the oxidized ores on account of the difficulty in controlling the poisonous dust, in addition to the expense of grinding. It would involve a roasting cost. It would mean a more expensive plant, and a more extensive plant for the same capacity, due to the slower percolation of the fine material, which would require a much larger vat area of less depth to hold the tonnage required. It would mean a supply of cheap salt. A chemical precipitant would probably have to be used to recover the copper.

By leaching with sulphuric acid, the oxidized ore can be treated raw and need not be crushed finer than ½ in. or 2-mesh size, or perhaps coarser, as the experiments will show. Owing to the high oxidation of the surface ores and apparently small zone of partly oxidized ores, a large tonnage can be expected to give a comparatively high extraction by this method. Sulphuric acid can be made as a by-product at the roasting

[1] Stuart Croasdale: *Engineering and Mining Journal*, vol. lxxvi, No. 9, pp. 312 to 314 (Aug. 29, 1903); *Trans.*, xlvi, 378 (1913); *Mining Magazine*, vol. x, No. 3, pp. 200 to 204 (March, 1914).

plant of the Calumet & Arizona Co. at Douglas, or if the calcines are converted into sponge iron and used as a precipitant for copper, the low-grade pyrite from Bisbee can be cheaply transported and acid made at the mine. If electrolytic precipitation can be used, a small percentage of the acid consumed can be regenerated. This method of extraction will permit of less expensive plant construction for the oxidized ores.

On the other hand, sulphuric acid alone will extract none of the copper existing as sulphide, only half of the copper existing as cuprite, and may be indifferent about the silicate of copper. It will extract none of the precious metals. Owing to the character of the oxidation of the ore, already mentioned, considerable iron and alumina pass into solution as readily as the copper, which will seriously interfere with any form of electrolytic precipitation and render the amount of acid regenerated a doubtful asset when compared with the amount consumed. As an offset to this objection, some of the iron is in a ferric condition, which assists materially in the solution of the minerals mentioned above.

If acid is made at Douglas, it will have to be transported several hundred miles in tank cars, which will have to be returned empty. If acid is made at the mine from low-grade pyrite, a gain will be made in transportation, but the cost of an acid plant will be added to the cost of the leaching plant and the calcines will have to be utilized. Almost an entirely new plant will have to be constructed for treatment of the intermediate and sulphide ores.

The problem is clearly one in arithmetic as well as metallurgy, so laboratory experiments were made on both methods.

Laboratory Experiments

The ore furnished me for these experiments came from the earlier development of the property and the oxidized ores were from very near the surface, but were less kaolinized than those received later for the larger experiments.

The analyses of the ores were as follows:

	Oxidized Per Cent.	Sulphide Per Cent.
SiO_2	66.23	66.08
Fe	6.40	4.50
Al_2O_3	13.75	11.35
CaO, total	0.56	0.55
CaO sol. in dilute acid	0.31	0.15
S, total	0.19	2.09
S as sulphate	0.03	0.03
Zn	0.10	0.10
Cu, total	2.03	2.75
Cu sol. in 10 per cent. H_2SO_4	1.84	0.22
Au, oz. per ton	0.01	0.01
Ag, oz. per ton	0.22	0.18

On a 2 per cent. ore, a variation of 0.01 per cent. in the percentage of copper in the tailings makes a difference of 0.5 per cent. in the percentage of extraction. This must be kept in mind when comparing the results of these experiments, for several hundredths of 1 per cent. may be within the limits of error in sampling and chemical analyses.

Chloridizing Roast with Salt

Oxidized Ores.—The ore was crushed to 20 mesh. The experiments were conducted in an assay furnace at a low red heat or at estimated temperatures of 1,000° to 1,300° F. No attempt was made to condense the volatile copper chloride fumes.

The roasted ore was leached 24 hr. with a 1 per cent. sulphuric acid solution which was supposed to correspond to the solutions from the condensing towers at the Pennsylvania Salt Works, Philadelphia, where this method of copper extraction has been in use for a long time.

In a Wedge, or other multiple-hearth furnace, a particle of ore remains in the roasting or chloridizing atmosphere several hours. My first experiments were on variation in time of roasting, which ranged from 1 to 6 hr.

Sulphur was added, in the form of pyrite, in slight excess of that necessary to combine with the copper as CuS, or 1.2 per cent; 10 per cent. salt was added, which is twice the amount necessary to form the normal chloride of copper, but this might be of some advantage, as will be mentioned later.

At the end of 1 hr., 11.8 per cent. of the copper had volatilized and 65.8 per cent. had been rendered soluble in the roasted ore, which gave a total extraction of 77.6 per cent.

At the end of 6 hr., 69.0 per cent. of the copper had volatilized and 7.3 per cent. remained soluble in the roasted ore, which gave a total extraction of 76.3 per cent.

The tests, at 1-hr. intervals between these extremes, showed a gradual increase in the percentage of copper volatilized, with substantially the same total extraction.

In the next series of experiments the sulphur was varied from 1.2 to 2.5 per cent. The salt was kept constant at 10 per cent. and the time of roasting in each case was 1 hr.

With 1.5 per cent. sulphur the results were 15.5 per cent. volatilization of the copper, and 70.3 per cent. soluble copper in the roasted ore, or a total extraction of 85.8 per cent.; 2.1 per cent. sulphur gave 21.1 per cent. volatilization and 64.7 per cent. soluble copper, or a total extraction of 85.8 per cent. Both higher and lower sulphur contents gave lower total extractions.

In the third series of experiments the sulphur was kept constant at

1.5 per cent. and the time of roasting at 1 hr. The salt varied from 5 to 15 per cent.

The lowest volatilization and highest total extraction were obtained with 10 per cent. salt, which, as stated above, were 15.5 and 85.8 per cent., respectively.

The result with 5 per cent. salt was a volatilization of 46.2 per cent. and a total extraction of 71 per cent. This amount of salt is near the theoretical quantity necessary to form the normal chloride of copper. My previous work on the volatilization of metals as chlorides proved that, other conditions being equal, volatilization greatly increases as the relations between the metal, salt, and sulphur approach theoretical proportions.

The use of 15 per cent. salt gave 23.2 per cent. volatilization and a total extraction of 80 per cent. Other proportions of salt gave intermediate results.

The foregoing experiments gave the best results when the ore, containing a little more than 2 per cent. copper, was roasted 1 hr. with 10 per cent. salt and 1.5 per cent. sulphur.

As a final test, these factors were all raised as follows: salt, 12.5 per cent.; sulphur, 2 per cent.; time, 1.5 hr. The results were, 24.2 per cent. of the copper volatilized and 62.3 per cent. soluble, or a total extraction of 86.5 per cent., which is substantially the same as before. The tailings from this test were leached an additional 24 hr. with a 5 per cent. sulphuric acid solution containing 7 per cent. salt, which increased the extraction to 90 per cent. In all probability a total of 48 hr. leaching with the weaker acid would have accomplished the same result.

Sulphide Ores.—The average analysis of these ores has been given. The copper existed as chalcopyrite. The experiments were conducted in a manner similar to those on the oxidized ores. The ore was crushed to 20 mesh. No sulphur was added. The percentage of salt ranged from 10 to 15 per cent. and, the ore being higher grade, the time of roasting was extended to 1.5 and 2 hr., although this was probably unnecessary. Owing to difficulty in controlling the furnace, the temperatures used were higher than they should have been.

The results were:

	1	2	3
	Per Cent.	Per Cent.	Per Cent.
Salt used	10.0	12.5	15.0
Copper volatilized	68.8	44.1	56.0
Soluble copper	20.0	47.9	37.2
Total extraction	88.8	92.0	93.2

Leaching these tailings an extra 24 hr. with 5 per cent. sulphuric acid solution containing 5 per cent. salt, extracted no more copper. There was no difficulty in extracting all the soluble copper in the roasted ore with a 1 per cent. acid solution.

While the temperatures, as stated above, were higher than they should have been, these results confirm those made a number of years ago, when I found that copper is more readily volatilized from chalcopyrite by the chloridizing roast than from any other mineral.

Mixed Oxidized and Sulphide Ores.—The following experiments were conducted by the Wedge Mechanical Furnace Co., at Philadelphia, to determine the merits of a sulphating roast. Their report is as follows:

Test No. 1. The oxidized ore was ground to 20 mesh and the sulphide to 40 mesh. Two parts of sulphide were mixed with one of the oxide and the mixture was roasted 3.5 hr. at temperatures increasing from 550° to 1,180° F. The roasted ore was leached with 2° B. sulphuric acid solution. The extraction was 85.5 per cent.

Test No. 2. The same mixture of ores was roasted 3 hr., with 5 per cent. salt added, at temperatures ranging from 400° to 990° F. The roasted ore was leached with 2° B. hydrochloric acid solution, which gave an extraction of 94.4 per cent.

Test No. 3. The same ore mixture was used as before, and 5 per cent. salt was added. The mixture was roasted 2.5 hr. at temperatures from 500° to 860° F. The acid wash was 2° B. hydrochloric acid. The extraction was 96.0 per cent. of the copper.

Test No. 4. The preceding test was repeated with the addition of 2 per cent. pyrite (containing 47 per cent. sulphur) to the mixture. The ore was roasted 2.5 hr. at 600° to 900° F. and the roasted ore leached with 3° B. hydrochloric acid. The extraction was 93 per cent.

Test No. 5. Equal parts of oxidized and sulphide ore were used and 5 per cent. salt added. Other conditions being the same, 94 per cent. extraction was obtained.

Test No. 6. Both ores were ground to 40 mesh and mixed in equal parts with the addition of 5 per cent. salt and 2 per cent. pyrite. The extraction was 94 per cent.

Test No. 7. Test No. 6 was repeated, except that 3 per cent. pyrite was added instead of 2 per cent. The mixture was roasted 2 hr. at temperatures from 400° to 900° F. The extraction was 98 per cent. of the copper.

Summary of Results.—These experiments are sufficient to show that an extraction of 90 to 95 per cent. of the copper can be expected by a chloridizing roast on all the ores whether mixed or treated separately, but it would be preferable to mix them in order to secure the advantage of an excess of sulphur in the sulphide ores. If they are not mixed, from 1.5 to 2 per cent. sulphur will have to be added to the oxidized ores in the form of pyrite.

The ores will have to be crushed to 20 mesh, or finer.

The salt required will be 5 to 10 per cent.

The temperature of roasting should not be above 900° F. to avoid excessive volatilization.

Large furnaces, where air supply and temperatures are under better control and where the benefits of mass action can be obtained, should give as good if not better results than the laboratory. At low temperatures, the slight volatilization losses are easily recovered by scrubbing towers, together with sufficient acid for washing the ore.

Leaching Experiments

These were made only on the raw oxidized ores. The crushing varied from 8 to 2 mesh sizes. The experiments were conducted in glass and stoneware jars provided with small air-lift pumps for circulating the lixiviants. Both standing and circulating lixiviants were tested. The strength of acid solution varied from 3 per cent., or just enough to dissolve the copper in the ore, up to a 10 per cent. solution. Sulphuric acid was used. All references to this acid mean 100 per cent. H_2SO_4 and not commercial acid. Only enough lixiviant was used in each experiment to cover the ore.

The results were as follows on 24-hr. treatment:

8-Mesh Ore

	Standing Lixiviant		Circulating Lixiviant	
Per Cent. Acid in Lixiviant	Per Cent. Cu Extracted	Lb. Acid Used per Lb. Cu Dissolved	Per Cent. Cu Extracted	Lb. Acid Used per Lb. Cu Dissolved
3.0	46.8	3.2	48.7	3.3
4.0	54.7	3.3	71.0	2.7
4.7	56.1	2.8	85.2	3.2
10.0	81.8	2.8	88.6	3.5

4-Mesh Ore

	Standing Lixiviant		Circulating Lixiviant	
Per Cent. Acid in Lixiviant	Per Cent. Cu Extracted	Lb. Acid Used per Lb. Cu Dissolved	Per Cent. Cu Extracted	Lb. Acid Used per Lb. Cu Dissolved
3.0	42.9	2.4	70.4	2.0
4.0	44.3	2.4	74.4	2.3
4.7	48.3	2.3	82.3	2.4
6.3	58.2	2.3	80.8	2.8
10.0	78.4	2.2	85.7	2.8

These experiments showed that circulating the lixiviant increased the extraction of copper without seriously increasing the consumption of acid; that crushing to 8 mesh increased the consumption of acid without materially increasing the extraction of copper; and that increasing the

strength of the lixiviant increased the extraction of copper and also the consumption of acid when the lixiviant was circulated through the ore. The fine ore produced in crushing to 8 mesh seriously interfered with the percolation of solutions.

Further experiments were made on 4-mesh material by increasing the time of leaching, with the following results:

	5 Per Cent. Acid Lixiviant		10 Per Cent. Acid Lixiviant	
Time of Leaching, Hr.	Per Cent. Cu Extracted	Lb. Acid Used per Lb. Cu Dissolved	Per Cent. Cu Extracted	Lb. Acid Used per Lb. Cu Dissolved
24	82.3	2.4	85.7	2.8
48	83.7	2.6	90.1	2.6
72	84.2	2.7	87.6	4.2
96	86.7	...	89.1	4.0

Additional time with 5 per cent. acid lixiviant increased the extraction very slowly—probably from the fact that the acid was nearly neutralized. The 10 per cent. acid lixiviant always had plenty of free acid remaining after the copper was all dissolved.

By referring to the analysis of this ore, it will be seen that the copper soluble in 10 per cent. sulphuric acid solution is only 90.6 per cent. when the sample is crushed to pass 100-mesh screen, so that a 10 per cent. acid lixiviant on a 4-mesh material for 48 hr. extracted about all of the copper that could be expected.

The above experiment was repeated on ore crushed to pass 0.5-in. or 2-mesh screen. Only a 10 per cent. acid lixiviant was used. The results, given below, were as good as those obtained from finer crushing.

Time of Leaching	Per Cent. Cu Extracted
48 hr. without change of lixiviant	86.0
48 hr. Lixiviant changed after 24 hr.	87.5
72 hr. Lixiviant changed after 48 hr.	90.0
96 hr. Lixiviant changed after 48 hr.	87.0

As already stated, the oxidized ore contained some cuprite. When clean sulphuric acid is added to this mineral, one-half the copper is set free in metallic condition, which remains insoluble in the acid. As nearly as could be determined by averaging a number of analyses, the amount of copper thus set free was about 0.2 per cent. of the ore, or say 10 per cent. of the total copper.

Analyses of the lixiviants from a number of leaching experiments showed that a certain amount of iron passed into solution simultaneously with the copper. This averaged from one-third to one-half of the amount of copper in solution, or from 0.8 to 1.1 per cent. of the solution. Some-

times, with high iron and much kaolinized ores, it amounted to more than this. The iron was about equally divided as ferric and ferrous sulphates.

Using 10 per cent. acid, the lixiviant passing from a charge of leached ore contained from 4 to 6 per cent. free acid in addition to the copper, iron, alumina, etc.

This combination of circumstances enabled me to use the partly neutralized lixiviant from a leached charge of ore for the first lixiviant on a new charge of ore and thereby not only neutralize the free acid in the lixiviant and at the same time enrich it in copper, but have a solvent for the cuprite (or metallic coppper if formed) in the form of ferric sulphate. Since ferric sulphate was always present and the lixiviants were always circulated, it is scarcely possible that any cuprite could escape solution.

As soon as the first lixiviant was neutral, it was sent to the precipitating vats. If it was left on the ore beyond the neutral point, some of the copper was reprecipitated in the ore. A new 10 per cent. acid lixiviant followed the first and finished leaching the ore, which in turn was neutralized on a new charge of ore.

Experimental Plant Tests

The Chloridizing Roast

The chloridizing roast and the subsequent leaching of the ore with water and scrubbing-tower solutions is a well-established metallurgical process and has been in commercial operation for a number of years on 2 per cent. copper ores.

Laboratory experiments told what could be expected from this process and there is little danger of unlooked-for difficulties in larger-scale operations. The questions involved are therefore more of relative costs of construction and operation than of metallurgical experiment, so, for the time being, no experiments were conducted on a larger scale.

Acid Leaching

Percolation Experiments.—A pipe 10 in. in diameter and 18½ ft. in length was used for an experimental percolation vat. For convenience in testing the various columns of ore, it was made in sections. The lower end of this vat was closed with a plate in which there was a 3-in. opening which served as a discharge gate for the ore and also for pipe connections to measure the flow of solution. This could also be attached to a centrifugal pump for circulating the solutions to the top of the vat if desired and in this way the working conditions of a commercial plant

could be produced. Aside from a small screen to protect the valves, no filter was used in the vat. The ore itself corresponded to the gravel filter that was expected to be used in practice.

The height of any ore column in this vat was the same as that used in any commercial plant. The filtering area was 78.54 sq. in., or 0.545 sq. ft., which corresponded to the same section of a full-sized vat. The area of any commercial vat divided by this area and the result multiplied by the number of gallons per minute flowing through a given column of ore would give the approximate amount of solution that would have to be handled by circulating pumps in a large plant.

All percolation experiments were continued for three or four days if possible, in order to detect any irregularity in the flow of solution by the segregation of slimes or from other causes.

Since 8-mesh ore was found to be of no advantage in the laboratory leaching experiments, it was not used in these experiments.

The facilities at Douglas for crushing ore for experimental purposes at that time were very poor. There was no opportunity for stage crushing with intermediate screening. The ore was reduced so that it would nearly all pass a 2-mesh screen in one operation with one crusher and one set of rolls. This undoubtedly produced a different product than would be obtained in the usual practice.

With 4-mesh ore. A screen analysis of this material gave

Mesh	Per Cent.
4 to 8	30.6
8 to 16	22.4
16 to 20	8.4
20 and finer	38.6

Between 3 and 4 per cent. of this product was a colloidal slime that could be suspended in water. This slime assayed 4.5 per cent. copper.

Ore crushed to this size absorbed 11 per cent. of its weight in water or 26.4 gal. per ton. This amount of water will be lost in tailings.

The additional quantity of water required to fill the interstices and cover a given quantity of ore in the vat was 15 per cent. of its weight or 36 gal. per ton. This will be known as one volume of water, or solution, in the experiments which follow, and its complete replacement will represent one cycle in the circulation of lixiviant, or one washing of the ore with water.

Experiment No. 1. The ore was charged into water to get the coarser particles on the bottom and form a better filter bed. Height of ore column, 5 ft. Fine ore and slime soon settled on top of the charge and practically stopped the percolation.

Experiment No. 2. Ore was charged into the vat dry. Height of ore column, 5 ft. Water was added on top of ore. Rate of percolation, 1

gal. in 15 min. To the charge, 2.5 ft. of ore was added, making the total column 7.5 ft. Rate of percolation, 1 gal. in 15 min. Another 2.5 ft. of ore was added, making the total column 10 ft. Rate of percolation, 1 gal. in 15 min. At the end of 72 hr. the rate was 1 gal. in 23 min., after which it gradually decreased.

When the charge was drawn, the fine ore was found to be more or less segregated, which was perhaps due to the method of charging.

Experiment No. 3. The above experiment was repeated. With a 5-ft. column of ore the rate of percolation was the same, 1 gal. in 15 min. The ore column was increased at once to 10 ft., in which the rate of percolation was 1 gal. in 10 min. The rate remained the same when the column was increased to 12 ft.

Experiment No. 4. The ore was charged dry and the height of the column was 18 ft. Water was introduced into the bottom of the vat until the ore was covered, in order to remove the air more quickly. It was then circulated from the bottom of the vat to the top by means of a centrifugal pump. Rate of percolation was 1 gal. in 8 min. At the end of 72 hr. it was still 1 gal. in 8 min. The height of the ore column was then reduced to 12 ft. and at the end of 120 hr. the rate of percolation was 1 gal. in 9 min.

Experiment No. 5. Upward percolation. The ore was charged dry. Height of column, 12 ft. Water was introduced from the bottom of the vat and upward percolation was maintained with a 9-in. head of water above the ore from a stand pipe outside the vat. The rate of percolation was 1 gal. in 6 min.

With an ore column of 15 ft. and a 10-in. head of water above the ore, the rate of percolation was 1 gal. in 7.5 min.

With 2-mesh ore. The screen analysis was

Mesh	Per Cent.
2 to 4	40.0
4 to 8	25.8
8 to 16	14.2
16 to 20	6.0
20 and finer	14.2

The colloidal slime was about 2 per cent. of the ore.

This ore absorbed 6 per cent. of its weight or 14.4 gal. of water per ton.

The volume of water necessary to cover the saturated ore was 27.7 per cent. of its weight or 66.5 gal per ton.

Experiment No. 1. Height of ore column, 12 ft. Water was circulated from bottom of vat to top by means of pump. Rate of percolation was 1 gal. in 30 sec. At the end of 96 hr., it was 1 gal. in 35 sec.

Summary.—The height of the ore column with 4 and 2 mesh ore has

no retarding effect on the rate of percolation, unless the ore is very much disintegrated and produces a good deal of slime. If anything, the rate is increased with increased height. Vats can therefore be constructed with any convenient depth, but will probably not be practicable beyond 10 or 15 ft., depending of course upon the character of ore and method of crushing.

The ore should be charged dry or only moist.

Lixiviants can be introduced best from the bottom of the vat or down one side of the vat from the top of the ore. This allows the free escape of air and carbon dioxide gas and permits a rapid saturation of the ore and filling of the vat. After the ore is covered, the percolation of the lixiviant through the ore should be downward. This permits better control of solutions and the washing of the ore with the minimum quantity of water.

By keeping the ore covered with solution and having the pumping or circulating capacity a little above the rate of percolation, channels through the ore can be detected and remedied.

The rate of percolation in this experimental vat may appear slow, but when the proper calculation is made for a large vat, it will be found that considerable pumping capacity will be required to take care of the solution.

Leaching Plant Construction

To meet the conditions at Ajo, reinforced concrete seemed to offer many advantages for large-sized leaching vats. This material has been used for a long time at the works of the Pennsylvania Salt Co., at Natrona and at Philadelphia, for leaching small tonnages of chloridized copper ores. The solutions there are a mixture of sulphate and chloride of copper and contain from 1 to 2 per cent. of free acid. No protective coating or lining has been used for the concrete although the new vat constructed at Philadelphia is lined with chemical brick laid with cement.

The stronger acid solutions used in leaching the Ajo ores necessitated some protective covering for the concrete. A paint of this character is made by Toch Brothers. It is known as "R. I. W. No. 44" and is advertised to have the following properties:

"1. It stands heat up to the point of carbonization.

"2. It withstands sulphuric acid, 20° strength, for two months without showing the slightest sign of decomposition.

"3. It withstands the action of 15 per cent. sulphuric acid, 10 per cent. copper sulphate, at a temperature of 135° F., for six months, without showing any sign of decomposition. Withstands influence of caustic soda up to about 10 per cent. strength.

"4. Two coats will last for several years on the interior of stand pipes. Steel must be clean and dry before material is applied.

"5. Has a record of three years and four months on the interior walls, floors and

ceilings of storage battery rooms where the vapors of sulphuric acid are condensed twice a day. Three coats were applied.

"The dates of its longevity are simply minimum dates, as the material lasts much longer than our statements indicate."

The protection of these concrete vats was submitted to Toch Brothers and they recommended the following specifications:

"Make your concrete a 1 : 2 : 4 mix, using an aggregate graded up to 1½ in., adding 3 lb. of 'Toxement' for each bag of cement necessary to complete the operation. After the concrete has set up thoroughly dry, clean the surface and then apply one coat of R. I. W. No. 89, allow about 36 hr. for it to dry, then apply a second coat of R. I. W. No. 89. Care should be taken after the first coat is applied so that every particle of the surface is thoroughly and completely covered. R. I. W. No. 89 is constructed slightly different from R. I. W. No. 44 in order to meet your specifications."

To make a concrete vat acid proof, it must first be made waterproof. For this purpose they recommended "Toxement," a patented preparation of their own which is advertised to be "a colloidal double resinate and silicate of calcium and aluminum."

A second method of waterproofing concrete was obtained from *Bulletin* No. 46, U. S. Department of Agriculture. In this method crude petroleum or preferably residuum is mixed with the concrete as follows:

"For most purposes where damp proofing is required, 5 per cent. of oil based on the weight of the cement in the mixture is all that is necessary. For each bag of cement (weighing 94 lb.) 4.7 lb. or about 2½ quarts of oil are required.

"Mix the concrete in proportions of 1 part cement to 2 of sand and 4 of broken stone or gravel. Add water and thoroughly mix until no trace of oil is visible on the surface. If oil-mixed mortar is desired, prepare in the same manner without the gravel.

"The use of 5 per cent. of oil increases the time of the initial set by 50 per cent. and the time of the final set by 47 per cent.

'Concrete with 10 per cent. oil has 75 per cent. of the strength of plain concrete at 28 days. At the end of one year the strength of 1 : 3 mortar suffers but little from the addition of oil in amounts up to 10 per cent.

"Oil-mixed mortar containing 10 per cent. of oil is absolutely water tight under pressures as high as 40 lb. to the square inch. Oil-mixed mortar is effective as a waterproofing agent under low pressures when plastered on either side of porous concrete."

Tests were also made on the acid-resisting qualities of the following bricks, which might be used for lining concrete vats: Ordinary wire-cut shale brick from El Paso; "acid-proof tile" from El Paso; low-temperature fire brick from El Paso; "Star" fire brick from Pueblo, Colo.; silica brick from Pueblo, Colo.

These bricks were allowed to stand three months in 10 per cent. sulphuric acid solution; 5 per cent. sulphuric acid and 5 per cent. copper sulphate solution; and 10 per cent. copper sulphate solution (10 per cent. $CuSO_4 + 5H_2O$).

There was no apparent disintegration or injury to the physical structure of any of these bricks at the end of the test.

Chemical analysis of the solutions likewise showed that none of the bricks had been appreciably attacked by either acid or copper sulphate solutions. The solutions from the silica brick contained some gelatinous silica, which probably came from the decomposition of the sodium silicate used as a binder, but this did not weaken the brick.

Two concrete storage tanks were made for holding wash water, in order to bring it up to normal strength (about 10° B.) of acid to be used as a lixiviant. In practice it would be better to build these tanks of steel or wood and line them with lead, but in this experimental plant concrete was used to determine its efficiency as well as that of acid-proof paints. Both of these tanks were made of oil-mixed reinforced concrete.

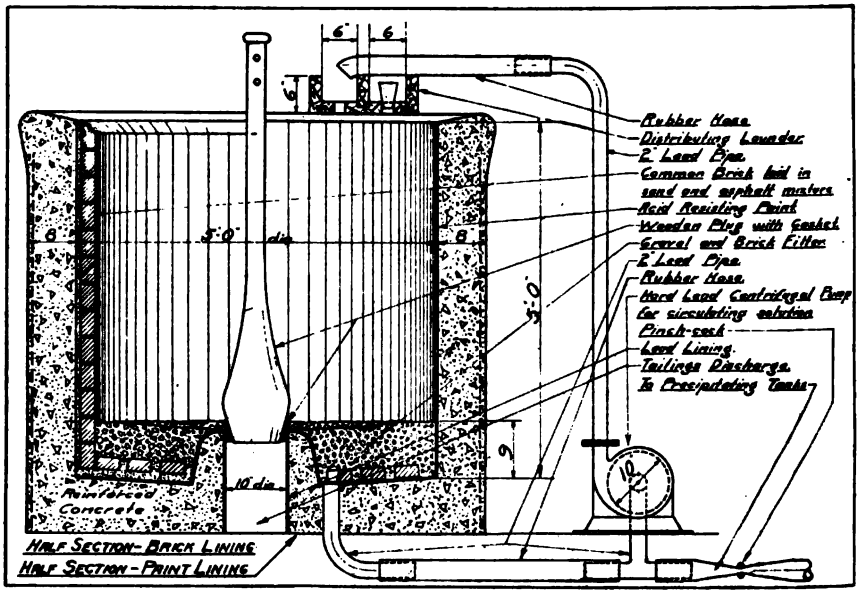


FIG. 1.—EXPERIMENTAL LEACHING TANKS.

They were 5 ft. in diameter and 5 ft. in depth, with walls 8 in. thick. One was plastered inside with "Toxement" mortar and the other with oil-mixed mortar. Both were painted inside with R. I. W. No. 89 paint as specified. For better protection from abrasion, the bottoms of these tanks were covered with ½ in. or more of hot asphalt and sand, troweled smooth and compact with a hot iron like street paving.

Five leaching vats were constructed according to the general design shown in Fig. 1.

For convenience in subsequent work these vats were designated as *A*, *B*, *C*, *D*, and *E*, and their individual construction was as follows:

Vat *A*. Five feet in diameter and 5 ft. in depth. Concrete, mixed

with "Toxement" according to the Toch specifications. Plastered inside with cement mortar mixed with "Toxement." Painted inside with three coats of R. I. W. No. 89 paint. A false lining of wood was made to set inside this vat, but it was impracticable to retain this lining without scarring the paint, so it was abandoned at the start. It was also impossible to keep the paint from being worn off by the ore, so the vat was given one coat of asphalt dissolved in gasoline and then lined with El Paso common brick. The bricks were laid in hot asphalt, and hot asphalt and sand were poured between the brick lining and the concrete. The bottom was covered with asphalt and sand, troweled smooth while hot.

Vat *B*. Same construction and dimensions as Vat *A*. This was given two coats of hot asphalt, which was difficult to apply and on cooling it contracted and left pin holes of uncovered cement. A thin paint was then made by pouring hot asphalt into gasoline and a coat of this was applied to the above. The bottom was covered with asphalt and sand, troweled smooth while hot.

Vat *C*. Same dimensions as Vats *A* and *B*, but made of oil-mixed concrete and plastered inside with oil-mixed cement mortar. A priming coat of R. I. W. No. 89 paint was applied to the concrete. On top of this was placed a coat of hot asphalt followed by a coat of asphalt paint as described under Vat *B*. The bottom was made of hot asphalt and sand, as already described.

Vat *D*. This was rectangular, 5 by 8 ft. and 5 ft. in depth. It was made of oil-mixed concrete and plastered inside with oil-mixed cement mortar. The protective coatings were one coat of R. I. W. No. 89 and two coats of asphalt paint as described above. The bottom was covered with hot asphalt and sand.

Vat *E*. Made of oil-mixed concrete and plastered inside with oil-mixed mortar. Diameter 5 ft. and depth 10 ft. Protective coatings, one of R. I. W. No. 89 and one of asphalt paint. Bottom covered with hot asphalt and sand.

The capacity of Vats *A*, *B*, and *C* was between 3 and 4 tons, respectively. Vats *D* and *E* each held a little more than 8 tons.

Three solution or precipitating tanks, 5 ft. in diameter and 5 ft. in depth, were made and protected in the same manner as Vat *E*.

Leaching-Plant Operation

These experiments were made on about 300 tons of oxidized ore shipped from the various shafts and test pits and during the development of the property.

Test 1.—This was made on two cars of ore, or Lots 1 and 2, which were crushed and mixed for treatment. As already stated, owing to

limited crushing facilities, it was necessary to crush with one crusher and one set of rolls, as much as possible to 2 mesh and finer in one operation. The result of this was only 7.25 per cent. of oversize and probably a greater production of fines than would be obtained in practice.

Lot 1 was hard unaltered ore similar to that used in the laboratory experiments. Lot 2 was considerably altered by weathering and produced a great deal of fine material and slime.

A screen analysis of the mixture gave the following results:

Mesh	Per Cent.
2 to 4	29
4 to 8	14
8 to 16	18
16 and finer	39

The chemical analyses of the two lots are given below:

	Dry Weight, Pounds	Copper, Per Cent.
Lot 1	62,172	1.84
Lot 2	94,570	1.37
	156,742	1.55

Chemical Analyses

	Lot 1 Per Cent.	Lot 2 Per Cent.		Lot 1 Per Cent.	Lot 2 Per Cent.
SiO_2	65.7	63.6	Cu, total	1.84	1.37
Fe	4.0	4.7	Cu sol. in 10 per cent. H_2SO_4	1.66	1.27
Al_2O_3	14.3	13.4	CO_2	1.70	1.33
				Ounce	Ounce
CaO sol. in acid	0.8	0.6	Au	0.01	0.01
S, total	0.3	0.4	Ag	0.20	1.18
S as sulphate	0.08	0.1			

The leaching was conducted in the manner described under laboratory experiments, namely, the partly neutralized lixiviant from finishing one charge of ore was neutralized and enriched by using it as the first lixiviant on a new charge of ore.

The details of this test are summarized as follows:

Total ore leached, pounds	125,857
Average copper contents, per cent	1.53
Average time of acid leaching, hours	70.6
Average time of washing, hours	5.0
Final tailing, total copper, per cent	0.30
Final tailing, soluble copper, per cent	1.12
Copper extracted per ton of ore, pounds	24.4
Per cent. extraction	80.0

Total weight of material extracted from the ore, per cent	3.5
Actual percentage of copper extracted	81.0
Percentage of available copper extracted	91.0
Lixiviant used per ton of ore, gallons	132
Average per cent. of acid in lixiviant	9.24
Acid consumed per pound of copper dissolved, pounds	3.4
Wash water used per ton of ore, gallons	138
Rate of percolation in inches per hour, measured in inches of solution above the ore in vat	90
Copper accounted for in solution, of the total amount extracted from the ore by difference between heads and tails, per cent	92.2
Average value of nearly neutralized lixivium sent to precipitating vats:	
Copper, per cent	2.22
Iron, per cent	0.42
Alumina, per cent	0.47

The average grade of the ore in this test was lower than that used in the laboratory tests. There is, however, not likely to be a corresponding decrease in value of the tailing from a 1.5 per cent. ore. below that obtained from a 2.0 per cent. ore, so the percentage of extraction decreases rapidly in proportion to the decrease in the grade of the ore.

The ore was charged dry into the vats. The lixiviants were admitted on top of the ore but at one side of the vat, and no faster than could be absorbed by the charge. By making a little dam on top of the charge, the lixiviant passed immediately to the bottom of the vat and saturated the charge from below, thus permitting the escape of air and gas.

The soluble copper in the tailing was determined by boiling the laboratory sample (100 mesh) in 10 per cent. sulphuric acid. The copper not soluble by this treatment must exist as sulphide or as other minerals in the ore, not capable of being extracted by sulphuric acid and therefore is not available by this metallurgical process.

Tests 2 *and* 3.—These were made on car lots 3 and 4 treated separately.

The ore was soft and much the same in character as Lot 2. It produced considerable fine material on crushing.

Lot 3 was crushed to ½-in. mesh.

Lot 4 under similar conditions crushed almost to ¼-in. mesh, as shown by screen analyses.

Mesh	Lot 3 Per Cent.	Lot 4 Per Cent.
−2+4	16	6
−4+8	27	27
−8+16	17	21
−16	40	46

	Dry Weight Pounds	Copper Per Cent.
Lot 3	111,230	1.48
Lot 4	89,670	1.44

Chemical Analyses

	Lot 3 Per Cent.	Lot 4 Per Cent.		Lot 3 Per Cent.	Lot 4 Per Cent.
SiO_2	64.00	63.30	Cu total	1.48	1.44
Fe	4.20	4.20	Cu sol. in 10 per cent.		
Al_2O_3	14.30	15.10	H_2SO_4	1.32	1.30
CaO sol. in acid	0.45	0.60	CO_2	1.42	1.43
				Ounce	Ounce
S, total	0.20	0.20	Au	0.01	0.01
S as sulphate	0.04	0.06	Ag	0.19	0.17

The results from these tests are shown in the following table:

	Test 2	Test 3
Pounds ore leached	90,533	80,458
Per cent. copper	1.44	1.43
Average hours acid leaching	70	100
Average hours washing	7	23
Final tailing, per cent. total copper	0.27	0.40
Final tailing, per cent. soluble copper	0.23	0.38
Pounds copper extracted per ton of ore	23.3	20.6
Percentage of copper extracted	81.2	71.8
Total per cent. of material extracted from ore	3.0	3.0
Actual percentage of copper extracted	81.8	72.7
Percentage of available copper extracted	83.5	73.0
Gallons lixiviant used per ton of ore	123	137
Average per cent. acid in lixiviant	9.5	7.0
Pounds acid consumed per pound copper dissolved	3.5	3.3
Gallons wash water used per ton of ore	135	177
Rate of percolation in inches per hour, measured in inches of solution above the ore in vat	43	11
Copper accounted for in solution	Combined with Test 4.	
Average value of nearly neutralized lixivium sent to precipitating vats:		
Copper, per cent	2.41	2.90
Iron, per cent	0.56	0.70
Alumina, per cent	0.53	0.73

These results show very clearly the effect of finer crushing on the same grade and character of ore. Both of these lots were substantially the same character of ore in every respect. They were very much altered by oxidation and contained practically no insoluble copper in the tailing, although the ore itself showed 0.08 to 0.10 per cent., probably as cuprite, which dissolved in the ferric salts extracted from the ore.

The extraction on the 4-mesh material was 9.4 per cent. lower and the rate of percolation dropped to one-quarter of that on the 2-mesh material. Hence much more time was consumed in leaching and washing the ore, with poorer results. Even on the 2-mesh ore the rate of percolation was only half that of the harder ore in Test 1, showing the effect of increased kaolinization of the ore.

In order to determine the relative value of samples from the top, and other parts of the charge, and also to determine whether the charges were leaching uniformly or in channels, a number of samples were taken from different charges at each foot in depth. This was done by removing half the charge from the vat and sampling the standing face of the remaining half of the charge across each foot of vertical section at a point half way between the center plug and the side walls of the vat.

A general sample was also taken in this manner, to check, not only the sectional samples, but also the one-tenth portion of the entire charge which was quartered in the usual manner for a tailing sample.

These results show that the leaching is very uniform throughout the charge on evenly distributed 2-mesh material, and that samples taken from the top of the charge may be used for preliminary tailing.

Lot 3, Test 2	Charge 22, Per Cent. Cu	Charge 24, Per Cent. Cu	Charge 25, Per Cent. Cu
Sample			
First foot, top	0.26	0.35	0.25
Second foot	0.27	0.30	0.25
Third foot	0.27	0.23	0.30
Fourth foot	0.29	0.26	0.35
Average	0.27	0.28	0.29
One-tenth portion of charge	0.26	0.27	0.26

Similar samples from the 4-mesh material (Car Lot 4, Test 3) did not give as uniform results. This contained not only much more fine material, but about 5 per cent. colloidal slime, as compared with 2 per cent. colloidal slime in the hard ore. Even though the ore was charged dry, this fine material was found to be irregularly segregated from the charging and apparently not from the action of the lixiviants. This segregation produced irregularities in the extraction by interference with the percolation and washing.

The results from the sectional sampling are given below:

Sample	Charge 26, Per Cent. Cu	Charge 27, Per Cent. Cu	Charge 28, Per Cent. Cu	Charge 29, Per Cent. Cu	Charge 32, Per Cent. Cu
First foot, top	0.24	0.31	0.23	0.46	0.24
Second foot	0.21	0.38	0.44	0.75	0.44
Third foot	0.33	0.40	0.27	1.10	0.34
Fourth foot	0.34	0.33	0.78	0.83	0.35
Fifth foot	.....	.....	.....	0.31	.....
Sixth foot	.....	.....	.....	0.44	.....
Seventh foot	.....	.....	.....	0.46	.....
Eighth foot	.....	.....	.....	0.31	.....
Ninth foot	.....	.....	.....	0.75	.....
Average	0.28	0.35	0.43	0.60	0.35
General	0.24	0.28	0.37	0.44	0.35
One-tenth portion of charge	0.26	.....	0.37	.....	.....

The soluble copper in all of the tailings from these two lots of ore shows its weathered condition by the almost total absence of copper as cuprite or sulphide.

The difficulty in extracting this copper led to a more detailed analysis of the tailings to determine whether the copper was retained by the coarse material alone, or whether the colloidal slime also had a tendency to absorb and retain copper even after it had gone into solution.

These results are interesting since they show that the colloidal slime absorbs, or retains mechanically, as much copper as is held by the coarse material. As anticipated, there is also an increase in the percentage of alumina in the fine material.

Screen and Chemical Analyses of Charge 25, *Lot* 3, *Test* 2

Size of Material, Mesh	Per Cent. of Weight	Per Cent. of Value	Per Cent. of Copper	Per Cent. Sol. Copper	Per Cent. of Silica	Per Cent. of Alumina
+4	9.3	15.2	0.47	0.39	68.0	12.6
+8	32.5	30.6	0.27	0.20	69.6	12.7
+16	23.2	19.5	0.24	0.18	68.5	13.9
+30	14.0	10.8	0.22	0.17	66.0	14.0
+60	4.7	4.3	0.26	0.19	66.9	13.5
+100	7.0	6.6	0.27	0.21	66.3	13.1
−100	9.3	13.0	0.40	0.37	63.3	16.4
	100.0	100.0	0.28	0.22	67.7	12.6

Screen and Chemical Analyses of Charges 29 *and* 30, *Lot* 4, *Test* 3

	Charge 29			Charge 30		
Size of Material, Mesh	Per Cent. Weight	Per Cent. Cu	Per Cent. Sol. Cu	Per Cent. Weight	Per Cent. Cu	Per Cent. Sol. Cu
+4	11.8	0.65	0.65	7.5	0.62	0.61
+8	25.9	0.54	0.53	24.6	0.47	0.46
+16	16.0	0.51	0.50	15.1	0.55	0.55
+30	16.8	0.54	0.54	17.0	0.58	0.58
+60	7.0	0.59	0.58	9.4	0.65	0.64
+100	8.0	0.59	0.59	15.1	0.70	0.70
−100	14.5	0.57	0.57	11.3	0.77	0.77
Average.....	100.0	0.56	0.55	100.0	0.60	0.59
General samples	.....	....	0.44	.....	0.42	....

This retention of the copper by the slime was also noticeable in washing the ore. Although the charges in Lot 4, Test 3, were washed a much longer time and with 50 per cent. more water than those in the previous tests, the analyses of the wash waters showed a very slow removal of the copper. Where the percolation was slow and the tailings high, even the sixth volume of wash water contained a high percentage of copper, while the acid was removed as readily as in previous tests. This confirms the well-known properties of colloidal or argillaceous material for retaining mineral salts.

The rate of percolation in Test 3, Lot 4, dropped to one-eighth of the rate in Test 1.

The acid consumption in these two tests was substantially the same as in Test 1. The average strength of the acid used in Test 3 was 7 per cent. as compared with 9.25 per cent. and 9.5 per cent. in the preceding tests. In this test partly neutralized lixiviants were used on new ore in every charge, which could not always be done in the other tests. This had a tendency to lessen the consumption of acid, and neutral or nearly neutral solutions could be sent to the precipitating tanks. As stated before, if the acid contents of these lixiviants are too low and they become completely neutralized, part of the copper is reprecipitated in the ore. It would therefore be advisable to send the lixivium to the precipitating tanks slightly acid. Lixiviums containing relatively high percentages of ferric sulphate, and perhaps aluminum sulphate in the form of alum, show an acid reaction that corresponds to 0.2 per cent. to 0.5 per cent. of sulphuric acid.

Test 4.—This was made on Car Lot 5.

Dry weight, pounds	96,924
Per cent. copper	1.20

This ore was hard and contained more sulphides and cuprite than the previous lots except Lot 1.

Chemical Analysis

	Per Cent.		Per Cent.
SiO_2	65.50	Cu, total	1.20
Fe	3.90	Cu, sol. in 10 per cent. H_2SO_4	1.04
Al_2O_3	14.60	CO_2	0.59
CaO sol. in acid	0.80		Ounce
MgO	1.15	Au	0.01
S, total	0.44	Ag	0.17
S as sulphate	trace		

The ore treated was screened through ½-in. or 2-mesh screen. Only 27.5 per cent. was finer than 16 mesh as compared with 40 per cent. in the preceding tests.

Screen Analysis

Mesh	Per Cent. Weight		Per Cent. Copper
+4	26.2		1.04
+8	30.3		1.11
+16	16.0		1.28
+30	11.3		1.40
+60	5.5	27.5	1.07
+100	5.0		1.92
−100	5.7		1.94
	100.0		1.37

The results from leaching this material are shown in the following table.

The oversize from Lots 3, 4 and 5, amounting to 4 per cent. of the total ore treated and ranging from ½ to 1 in. in size, was recrushed to pass a 2-mesh screen and leached separately. This oversize was substantially the same in character as the finer material but had passed through the rolls uncrushed due to the irregular hand feeding.

A screen analysis of the recrushed oversize gave the following results:

Mesh	Per Cent.
+4	32.0
+8	26.3
+16	14.7
−16	27.0

The leaching tests on this product are also shown in the following table:

	Test 4	Recrushed Oversize Tests 2, 3 and 4
Pounds ore leached	77,636	10,300
Per cent. copper	1.29	1.49
Average hours acid leaching	77	90
Average hours washing	10	10
Final tailings, per cent. total copper	0.32	0.24
Final tailings, per cent. soluble copper	0.16	0.16
Pounds copper extracted per ton of ore	19.3	25.2
Percentage of copper extracted	75.0	83.8
Total percentage of material extracted from the ore	3.0	3.0
Actual percentage of copper extracted	75.8	84.4
Percentage of available copper extracted	82.9	91.2
Gallons lixiviant used per ton of ore	120	145
Average per cent. acid in lixiviant	9.3	10.8
Pounds acid consumed per pound of copper dissolved	4.3	3.8
Gallons wash water used per ton of ore	666	712
Rate of percolation in inches per hour, measured in inches of solution above ore in vat	161	75
Per cent. of copper accounted for in solution, of the total amount extracted from the ore by difference between heads and tails. (Combined solutions from Tests 2, 3, and 4)	99.8	.....
Average value of nearly neutralized lixivium sent to precipitating vats		
Copper, per cent	2.30	.....
Iron, per cent	0.56	.....
Alumina, per cent	0.66	.....

The lower extraction in this test is due to the lower grade of the ore and the high percentage of sulphides in the ore. I have invariably

found in experimenting with low-grade ores that the value of the tailing is a very constant factor, whether the value of the ore be $5 or $10 per ton.

In the analyses of the tailings in Tests 2 and 3 we found that practically all of the copper was soluble copper and that the extremely fine material carried more copper than some of the coarser sizes.

In this test several charges of the leached ore were washed with as much as 1,500 gal. of water per ton, or until the final wash water showed no copper or acid, which accounts for the high average volume of wash water in the results given above. This extra and complete washing had absolutely no effect on the soluble copper in the tailings, which remained as high as in charges washed with one-tenth of the water.

The following screen analyses on the tailings from two of these charges show that even the finest material retains copper soluble in sulphuric acid that will not wash out. Charge 41 came from *E* vat, 9 ft. in depth, and Charge 43 came from *A* vat, 4 ft. in depth.

	Charge 41			Charge 43		
Size of Material Mesh	Per Cent. Weight	Per Cent. Total Cu	Per Cent. Sol. Cu	Per Cent. Weight	Per Cent. Total Cu	Per Cent. Sol. Cu
+4	27.8	0.30	0.23	23.5	0.32	0.24
+8	30.9	0.27	0.20	34.7	0.17	0.11
+16	16.6	0.32	0.21	16.5	0.12	0.08
+30	10.8	0.34	0.25	9.5	0.18	0.13
+60	4.6	0.49	0.30	5.2	0.34	0.19
+100	3.6	0.50	0.33	4.3	0.38	0.19
−100	5.7	0.55	0.36	6.3	0.55	0.32
Totals	100.0	0.33	0.23	100.0	0.24	0.16
Original tailings	.....	0.30	0.13	.....	0.24	0.15

This fact points to the importance of studying carefully the gangue of an ore, even if it may not be acid consuming. The ability of argillaceous and some colloidal material to absorb and retain mineral salts is well known and these experiments have shown it to be quite a factor in leaching raw oxidized porphyry ores with sulphuric acid.

An effort was made to determine the rate of extraction of copper from the ore by the several lixiviants. To do this the ore and lixiviant were sampled every 2 hr. The ore samples were taken from the top of the charge and the results obtained from them were not entirely satisfactory, but were the best obtainable.

The first charge tested in this way was Charge 20, Test 2. Two strong lixiviants were used on this charge—the first containing 14.2 per cent. acid and the second, 12.7 per cent. acid. The results are shown in the following table and are graphically presented in Fig. 2.

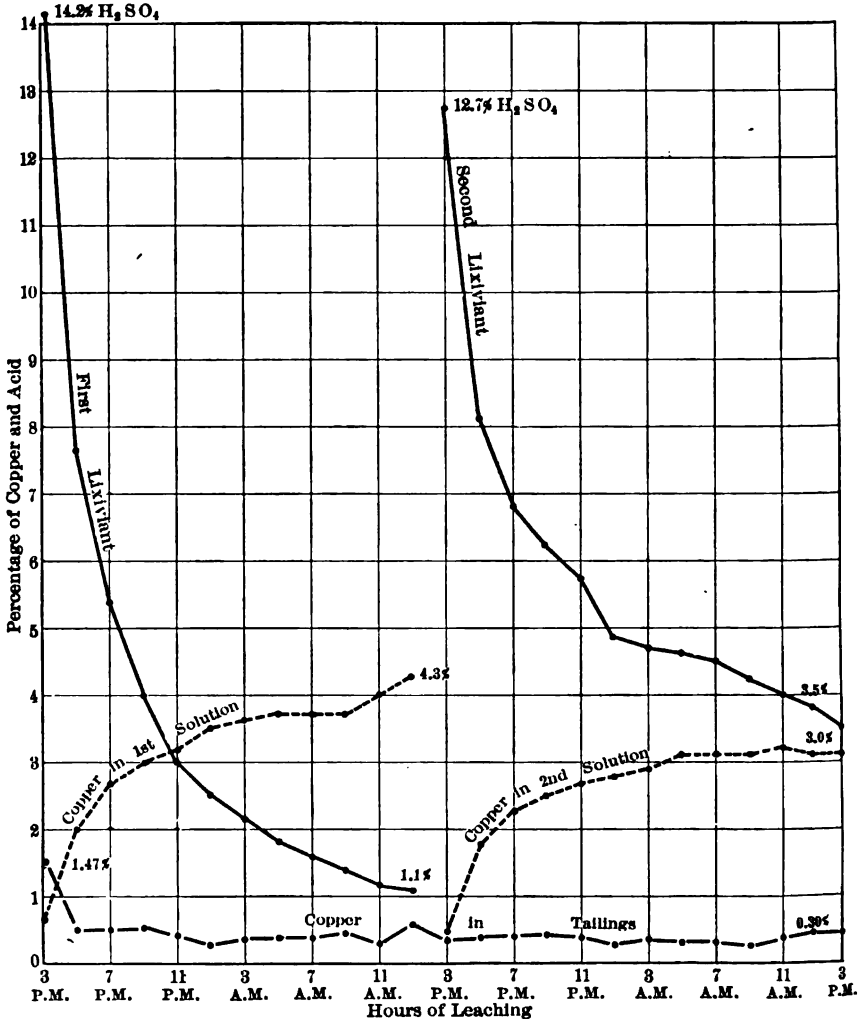


FIG. 2.—RELATIVE VARIATIONS IN COPPER AND ACID CONTENTS OF LIXIVIUM AND ORE, CHARGE 20, VAT *D*.

Leaching Experiments on Charge 20, Vat D. Relative Variations in Copper and Acid Contents of Lixivium and Ore

Sample Time	Ore Per Cent. Cu	Solution Cu, Per Cent.	Solution H_2SO_4, Per Cent.
First Lixiviant			
3 p.m.	1.47	0.7	14.2
5 p.m.	0.47	2.0	7.7
7 p.m.	0.55	2.8	5.4
9 p.m.	0.55	3.0	4.0
11 p.m.	0.38	3.2	3.0
1 a.m.	0.28	3.5	2.5
3 a.m.	0.35	3.6	2.2
5 a.m.	0.35	3.7	1.8
7 a.m.	0.35	3.7	1.6
9 a.m.	0.41	3.7	1.4
11 a.m.	0.30	4.0	1.2
1 p.m.	0.59	4.3	1.1
Second Lixiviant			
3 p.m.	0.31	0.4	12.7
5 p.m.	0.31	1.7	8.1
7 p.m.	0.32	2.2	6.8
9 p.m.	0.32	2.5	6.2
11 p.m.	0.28	2.7	5.7
1 a.m.	0.32	2.9	4.8
3 a.m.	0.28	2.8	4.7
5 a.m.	0.27	3.1	4.6
7 a.m.	0.25	3.1	4.5
9 a.m.	0.35	3.1	4.2
11 a.m.	0.29	3.2	4.0
1 p.m.	0.29	3.1	3.8
3 p.m.	0.34	3.0	3.5

The next test was made on Charge 38, Test 4. This is more complete and shows the relative variations in the copper and acid contents of lixiviants and ore and also the quantities of iron and alumina passing into solution.

The first lixiviant used here was a partly neutralized one from a previous charge. This was followed by two new lixiviants, each containing 10 per cent. acid.

Time of Sample	Per Cent. Cu in Ore	Lixiviums H_2SO_4 Per Cent.	Cu, Per Cent.	Al_2O_3 Per Cent.	Fe (Total) Per Ct.	Fe (Ferric) Per Ct.	Fe (Ferrous) Per Ct.
First Lixiviant							
1 p.m.	1.30	4.00	1.40	.....	.....	.....	.....
3 p.m.	0.99	1.99	2.06	0.66	0.83	0.50	0.33
5 p.m.	1.02	1.39	2.27	0.70	0.80	0.41	0.39
7 p.m.	.....	0.81	2.40	0.74	0.82	0.44	0.38
9 p.m.	1.10	0.34	2.52	0.74	0.80	0.40	0.40

Time of Sample	Per Cent. Cu in Ore	Lixiviums: H_2SO_4 Per Cent.	Cu, Per Cent.	Al_2O_3 Per Cent.	Fe (Total) Per Ct.	Fe (Ferric Per Ct.	Fe (Ferrous) Per Ct.
Second Lixiviant	.....	10.31	0.90	0.57	0.62	0.49	0.13
11 p.m.	0.56	6.52	1.68	0.57	0.63	0.30	0.33
1 a.m.	0.61	4.74	2.18	0.61	0.72	0.38	0.34
3 a.m.	0.53	3.51	2.40	0.65	0.71	0.35	0.36
5 a.m.	0.64	3.02	2.48	0.67	0.72	0.33	0.39
7 a.m.	0.36	2.58	2.52	0.61	0.86	0.45	0.41
9 a.m.	0.35	2.13	2.54	0.77	0.85	0.43	0.42
11 a.m.	0.38	1.78	2.61	0.79	0.81	0.42	0.39
1 p.m.	0.36	1.48	2.69	0.87	0.76	0.34	0.42
3 p.m.	0.28	1.32	2.78	0.77	0.87	0.43	0.44
5 p.m.	0.26	1.01	2.90	0.87	0.89	0.38	0.51
7 p.m.	0.28	0.82	2.99	0.89	0.88	0.38	0.50
9 p.m.	0.29	0.72	3.00	0.94	0.89	0.40	0.49
Third Lixiviant	.....	10.00	0.40	.....	0.42	0.16	0.26
11 p.m.	0.32	6.35	1.38	0.58	0.60	0.28	0.32
1 a.m.	0.47	4.70	1.84	0.62	0.67	0.31	0.36
3 a.m.	0.29	3.88	2.02	0.67	0.70	0.29	0.41
5 a.m.	0.38	3.44	2.03	0.76	0.74	0.33	0.41
7 a.m.	.....	3.06	2.32	0.77	0.83	0.42	0.41
9. a.m.	0.28	2.68	2.40	0.87	0.87	0.46	0.41
11 a.m.	0.31	2.30	2.47	0.89	0.88	0.47	0.41
1 p.m.	0.24	2.00	2.54	0.93	0.96	0.53	0.43
3 p.m.	0.29	1.80	2.54	0.96	0.98	0.43	0.55
5 p.m.	0.35	1.64	2.54	0.99	1.00	0.49	0.51
7 p.m.	0.32	1.48	2.54	1.03	1.04	0.51	0.53
9 p.m.	0.33	1.44	2.54	0.98	1.09	0.55	0.54

The percentage of copper dissolved from the ore by the several lixiviants in this experiment, as accounted for by the copper in solution, was as follows:

	Hours on Ore	Pounds Copper Taken into Solution	Per Ct. of the Total Copper Extracted
First lixiviant	8	31	23.7
Second lixiviant	24	34	25.9
Third lixiviant	24	51	38.9
Wash water	7	15	11.5
Totals	63	131	100.0

The lixiviants slowly dissolved iron and alumina from the ore, reaching a maximum of about 1 per cent. of each constituent. About one-half of the iron was in the ferric condition, which is of value in dissolving copper, as has already been explained.

These results are graphically shown by Fig. 3.

In the foregoing tests the acid consumption was nearly 3.5 lb. instead of under 3 lb. as obtained in the laboratory experiments. While this was undoubtedly due to the more decomposed ore used in the larger experiments, yet it might be possible to secure a selective action of the acid for the copper by using weaker lixiviants and leaching the ore a longer time. If this is true, less iron and alumina would pass into solution and less acid would be consumed.

The reject from the charge samples of Tests 1, 2, 3, and 4 amounted

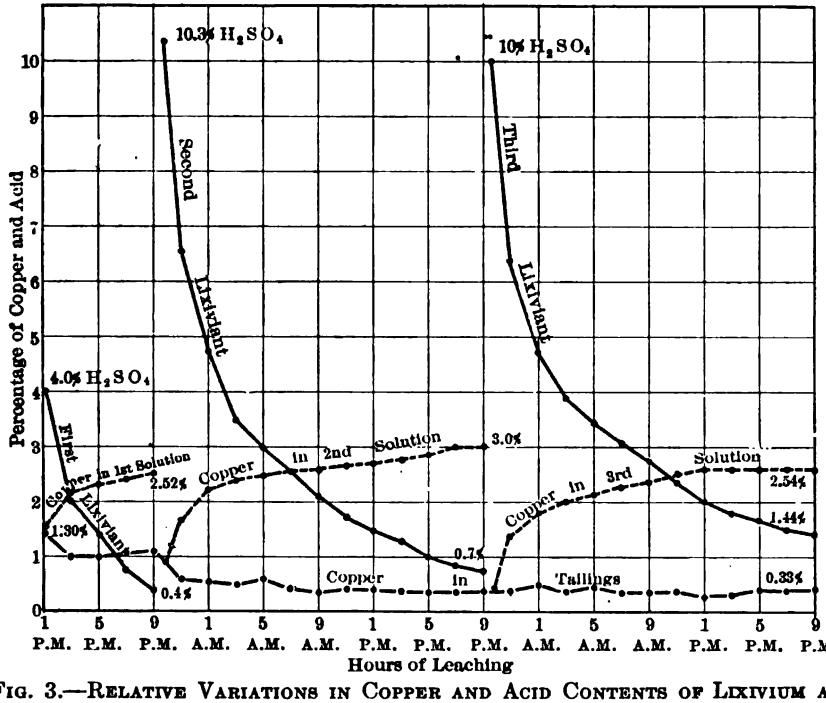


FIG. 3.—RELATIVE VARIATIONS IN COPPER AND ACID CONTENTS OF LIXIVIUM AND ORE, CHARGE 38, VAT *D*.

to 30,345 lb. In order to test the advisability of leaching the ore with weaker acid and taking more time for the operation, this reject was treated in the experimental plant with an average of 5.25 per cent. acid lixiviant. The first lixiviant used on each charge was the partly neutralized lixiviant from a previous charge that had been treated with a 10 per cent. acid, and already contained 1.16 per cent. copper. Each charge was also saturated with water before putting on the lixiviant, to determine if this would effect any saving in acid.

The results were:

Vat.	Charges			Tailings			Extraction			Time of Leaching, Hours	Lixiviant Changed Every	Per Cent. Acid		Lb. Acid Used per Lb. Cu Dissolved
	Weight, Lb.	Cu, Per Cent.	Cu, Lb.	Total Cu, Per Cent.	Sol. Cu, Per Cent.	Total Cu, Lb.	Total Cu, Lb.	Cu, Lb. per Ton	Per Cent.			Going On	Going Off	
A.....	6,020	1.42	85	0.72	0.60	43	42	14.0	49.5	96	24 hr.	5.50	tr.	3.0
B.....	7,825	1.40	109	0.32	0.23	25	84	21.5	77.0	120	24 hr.	5.20	tr.	2.0
D.....	16,500	1.76	290	0.40	0.36	66	234	28.4	77.2	144	24 hr.	5.10	tr.	2.0
Total...	30,345	1.60	484	0.44	0.37	134	350	23.0	72.3	120	.....	5.25	tr.	2.1

The relative percentages of copper, iron, and alumina dissolved by the several lixiviants on each charge are shown below. Owing to the incomplete extraction and shorter time of leaching on Vat *A*, those results are not comparable and are omitted.

Vat *B*

Extracted by	Hours on Ore	Per Cent. Acid		Relative Percentages Dissolved		
		Going On	Going Off	Cu	Fe	Al_2O_3
First lixiviant.....	24	5.60	trace	1.0	0.0	0.0
Second lixiviant.....	24	5.40	trace	20.3	31.9	17.6
Third lixiviant.....	24	5.40	trace	22.1	21.3	18.9
Fourth lixiviant.....	24	4.79	trace	28.3	21.3	29.7
Fifth lixiviant.....	24	4.79	trace	27.3	25.5	33.8
		5.20		100.0	100.0	100.0

Vat *D*

Extracted by	Hours on Ore	Per Cent. Acid		Relative Percentages Dissolved		
		Going On	Going Off	Cu	Fe	Al_2O_3
First lixiviant.....	24	5.60	trace	1.0	0.0	0.0
Second lixiviant.....	24	5.40	trace	16.4	18.2	8.8
Third lixiviant.....	24	5.40	trace	19.0	10.2	15.5
Fourth lixiviant.....	24	4.79	trace	21.9	21.6	26.3
Fifth lixiviant.....	24	4.79	trace	20.9	21.6	25.5
Sixth lixiviant.....	24	4.79	trace	20.2	28.4	23.9
		5.10		100.0	100.0	100.0

The ore being saturated with water in each of these charges, each lixiviant put on represented one volume of solution, or enough to fill the interstices and cover the charge. The results in each case show that the first lixiviant, containing 5.60 per cent. acid, was but little more than enough to destroy the alkalinity of the ore and very little of the copper was dissolved. In fact, the first lixiviant coming from Vat *A* contained less copper than it did going on, showing that the acid was not sufficient to destroy the alkalinity of the ore and some of the copper was reprecipitated in the ore. This confirms the results already obtained in the experimental plant, where there was usually very little increase and sometimes a considerable decrease in the copper contents of the first lixiviant if the free acid and ferric sulphate in solution were not sufficient to destroy the alkalinity of the ore. Therefore it is a waste of time and expense to use lixiviants too weak to accomplish this purpose and to bring about a maximum solution of copper with the first lixiviant.

On the other hand, where a 10 per cent. or stronger acid was used, 75 or 90 per cent. of the total copper dissolved can be extracted by the first lixiviant in 36 to 48 hr.

Each succeeding lixiviant dissolved about the same amount of copper and a corresponding amount of iron and alumina. As in the laboratory tests, iron dissolved relatively as rapidly as the copper while the alumina dissolved in increasing proportions with each succeeding lixiviant.

The final lixiviants on each charge were made from new acid.

That the extraction was not completed, even at the end of the sixth day on Vat *D*, is shown by the complete neutralization of the acid in the final lixiviums and no decrease in their copper contents. Had the leaching been continued to the complete extraction of the copper, the increasing percentage of alumina, together with the iron passing into solution, would no doubt have increased the acid consumption to approximately that already obtained in the experimental plant.

It was found inadvisable to saturate the ore with water before applying the first lixiviant. This water must, of course, be replaced by the acid before the copper can be dissolved and the interchange takes place but slowly through the pores of the ore.

The acid lixiviants should be applied to the dry ore.

Summary of Operations

Analysis of a composite sample of the five lots of ore treated gave the following results.

The average grade of this entire quantity of ore was much lower than anticipated and lower than that used for the laboratory experiments.

All of the ore treated passed through a 2-mesh screen except the oversize from Test 1, which amounted to 9,818 lb. This oversize

ranged between 0.5 and 1 in. in size and was treated by itself as a special experiment to conform more closely to the practice of the Arizona Copper Co. at Clifton.

	Calculated from Five Samples, Per Cent.	Composite Sample, Per Cent.		Calculated from Five Samples, Per Cent.	Composite Sample, Per Cent.
SiO_2	64.27	63.29	Cu, total	1.44	1.45
Fe	4.18	4.20	Cu sol. in 10 per cent. H_2SO_4	1.31	1.32
Al_2O_3	14.73	14.30	CO_2	1.30	1.26
CaO sol. in acid	0.63	0.90		Ounce	Ounce
MgO	.....	0.80			
MnO	.....	0.14	Au	0.01	0.01
S, total	0.32	0.27	Ag	0.18	0.18
S as sulphate	0.05	0.10			

The first charge was leached only 20 hr. and gave an extraction of 51.7 per cent. The second charge was leached 39 hr. and gave an extraction of 71 per cent.; making an average leaching time of about 30 hr. and yielding an average extraction of 61.3 per cent. If the time of leaching had been prolonged to 70 hr., corresponding to that of the 2-mesh material, the extraction of the whole lot would undoubtedly have been as good as that of the 2-mesh material alone, for by combining the results we have:

Size of Material,	Per Cent. Weight	Extraction	
		Lb. Cu per Ton	Per Cent.
1 to 0.5	7.25	21.6	61.3
0.5 and finer	92.75	24.4	80.0
Total	100.00	24.2	78.4

Therefore in all probability the ore can be leached successfully if not crushed finer than will pass a 0.75-in. screen, which will avoid much of the fine material and slime.

The ore in Test 3, on the other hand, nearly all passed a 4-mesh screen. The fine material produced in crushing to this size very seriously interfered with the percolation of the lixiviants, and washing the ore; consequently it caused loss of time and gave low extractions.

Had the entire quantity of ore treated been thoroughly mixed and crushed to a uniform size, the results would have been better than the average of the results obtained by treating each lot separately, so the

average results given below are probably not the best obtainable and are certainly on the safe side.

Screen Analysis

Mesh	Per Cent.
+4	19
+8	25
+16	18
−16	38
	100

Total ore treated, pounds	384,784
Copper in ore, per cent	1.43
Total copper in ore, pounds	5,526
Available copper in ore (soluble in H_2SO_4 and lixiviant), pounds	5,097
Time of leaching, hours	80
Time of washing, hours	10
Copper in tailing, per cent	0.32
Total copper in tailing, pounds	1,228
Soluble copper in tailing, per cent	0.21
Total soluble copper in tailing, pounds	799
Copper in tailing insoluble in H_2SO_4, pounds	429
Total copper extracted from ore, pounds	4,298
Total copper extracted from ore, per cent	77.8
Copper accounted for in solution including that from oversize in Test 1, pounds	4,269
Copper accounted for in solution including that from oversize in Test 1, per cent	97
Total copper extracted from ore based on weight of tailing at 97 per cent. of ore, pounds	4,336
Total copper extracted from ore based on above weight of tailing, per cent.	78.5
Soluble or available copper extracted from the ore, per cent	84.3
Soluble or available copper extracted from the ore based on above weight of tailing, per cent	84.7
Copper extracted per ton of ore, pounds	22.34
Copper extracted per ton of ore based on above weight of tailing, pounds	22.50
Average strength of acid used, per cent	8.9
Acid used (100 per cent. H_2SO_4) per pound of copper dissolved, based on the difference in free acid in the lixiviants going on and going off each charge, as found by analyses, pounds	3.6
Acid used (100 per cent. H_2SO_4) per ton of ore on above basis, pounds.	80
Acid used (100 per cent. H_2SO_4) per pound of copper dissolved, based on the total combined acid found by analyses in all the solutions sent to precipitating tanks, wash waters, and lixiviants returned to storage, pounds	3.15
Acid used (100 per cent. H_2SO_4) per ton of ore on above basis, pounds.	67
Lixiviants used per ton of ore, gallons	128
Wash water used per ton of ore, gallons	268
Solution required to cover 1 ton of ore after saturation, equals one volume, gallons	34
Rate of percolation through ore, inches per hour	78
Rate of percolation per square foot of filter area, gallons per hour	10

Summary of Construction

Of the two methods recommended for waterproofing concrete viz., by the use of "Toxement" and by the use of crude oil, the concrete mixed with crude oil appeared to be less attacked by the acid solutions at the end of two months' service than that mixed with "Toxement," although the difference was scarcely noticeable.

Both methods require a much longer time for the concrete to set than when neither is used.

So far as could be observed, the acid-resisting qualities of concrete and plaster prepared with either "Toxement" or oil were not any better than those of ordinary concrete and cement plaster made with siliceous sand and gravel.

Acid-resisting paint known as "R. I. W. No. 89" prepared especially for this work was of no value whatever in protecting the concrete vats from the 10 per cent. sulphuric solutions, the weaker lixiviants in the leaching vats, or the neutralized copper sulphate solutions in the precipitating tanks.

Asphalt applied hot or as a paint was likewise useless as a protective covering for the cement plaster under any of the above conditions. One difficulty in retaining this on the walls of the vats was the high temperature of the water used for washing the ore, which averaged 100° F. or more.

The use of such warm wash water may be criticised and would have been changed under ordinary conditions, but there was a possibility of the water supply coming from hot springs or wells, if developed near the mine, so it was thought best to put the asphalt coverings to the severest test. Besides, a glance at the atmospheric temperatures during the summer months, as given in the beginning of this article, is sufficient to show that even this part of the experimental work required some care in manipulation. Asphalt that would soften and run from joints between bricks or from walls under the heat of the sun, might be brittle enough to crack at night and allow the acid solutions to get at the cement behind it.

With the exception of Vat *A*, all the leaching vats, acid and precipitating tanks may be considered as operated for the entire two months on the cement plaster alone, for at no time did the protective covering last more than 24 to 48 hr. on the leaching vats, and on the acid and precipitating tanks the paint would come off in patches after a few hours' or a few days' exposure to the solutions. At the end of two months the "Toxement" plaster, ½ in. thick, was disintegrated to the concrete in places on Vat *B* and the oil-mixed plaster of the same thickness was disintegrated nearly to the concrete on Vat C. In the acid storage tanks both the "Toxement" and oil-mixed plaster were dissolved in patches by the 10 per cent. acid solutions and the solutions leaked through the

8-in. concrete walls. These holes were patched with ordinary cement and sand plaster and also with straight cement, both of which held the solutions successfully for several weeks.

One of the storage tanks was replastered with cement and sand and thoroughly coated with paraffin, which was forced into the plaster with a hot iron. When filled with dilute acid lixiviant, the paraffin came off immediately and was of no value whatever as a protective covering.

A plaster, recommended by the Standard Oil Co., was also tried on the acid storage tanks. It was composed, by weight, of 10 per cent. litharge, 20 per cent. short fiber asbestos, and 70 per cent. sand. These were mixed into a mortar with 40° silicate of soda and applied in the usual manner.

This plaster immediately disintegrated on filling the tank with water. The tank was then replastered with the same materials and filled with dilute acid. So long as the plaster remained covered with acid, it was a satisfactory protection, but on exposure to air it began to disintegrate.

Some of the leaching vats were plastered with ordinary cement and sand mortar and painted with several coats of crude petroleum. Each application of oil was allowed to soak into the cement and become perfectly dry before another coat was applied. This was done at the close of my experiments and I had no opportunity to test its merits.

The brick lining in Vat *A* proved to be very satisfactory. With the exception of where the asphalt was melted from the joints by the hot wash water, no deterioration could be noticed. This was due to the irregular size and shape of the common brick used for the lining and could be avoided by using pressed brick of uniform size which would permit of thin joints. In a large vat the brick would also be laid flat instead of on edge.

The sand and asphalt bottoms were entirely satisfactory when they were pressed solid with a heavy hot iron muller like that used in street paving. If this was not done, they were porous and spongy, and no protection against the solutions.

As now developed by the limited operation of the experimental plant, ordinary or oil-mixed concrete will be entirely satisfactory for leaching vats, wash-water and neutralized copper sulphate solution tanks. These can be made tight and acid resisting by lining with pressed brick of uniform size, laid in asphalt and backed with hot asphalt poured between the brick and concrete. Vitrified brick would be less porous and preferable if they can be obtained straight and uniform in size. The vats should be plastered inside with cement and sand mortar before putting in the brick lining.

The cement we used in construction came from the regular run of

the mill at El Paso and contained more or less free lime. It is my belief that with cement and other materials containing *no* free lime, a concrete vat can be built and lined with brick, laid with thin joints of cement and sand mortar, that will be entirely satisfactory for acid leaching. After the brick soak full of mineral salts, there is very little if any transfusion of the corrosive solutions to the concrete back of the lining and they form a protective coating for both ore and solutions.

Oil-mixed cement plaster will last two or three months without patching or renewal. A brick lining laid in oil-mixed cement mortar would probably last longer than plaster, but the asphalt and brick lining is recommended if it can be held in place.

The cracks that form after construction are the most serious, if not a fatal defect in concrete for leaching vats. These are often so small that they are scarcely noticeable and yet are sufficient to start a leak that is almost impossible to stop. even with asphalt itself.

The shape of the vats is immaterial, although for economy of construction and operation a rectangular concrete vat is preferable. The tailings can then be removed from the top, and the bottom of the vat made solid with no openings.

The bottoms of the vats should be paved with sand and asphalt, thoroughly pressed down with hot iron mullers or otherwise to make a *compact* covering. On top of this should be a brick paving laid in asphalt.

The storage or acid mixing tanks for the 5 to 10 per cent. acid solutions should be made of steel or wood and lined with lead.

The hard-lead centrifugal pumps used for circulating the solutions showed no wear except the steel shafts. It was difficult to keep packing tight enough around the shafts to prevent the leakage of solutions, and at the same time not crack the lead casing or stuffing gland. The shafts on all the pumps were worn out in two months and were replaced with bronze. The wear on the bronze shafts is undetermined. The repairs on the pumps will not be a serious matter since new casings, runners and shafts are easily made.

Low-pressure air-lift pumps offer more advantages, on account of freedom from moving parts in the corrosive solutions, and are recommended.

The pure-rubber acid-proof hose used for the transfer of solutions showed no deterioration at the end of two months. While the initial cost is high, it is the only hose that will resist the corrosive solutions and it is indispensable for handling solutions from the pumps on top of the vats.

Lead-lined iron pipe is recommended for the rigid pipe connections and wooden, lead-lined or concrete launders provide the simplest means for the transfer of solutions from one leaching vat to another.

The tailings will have to be removed from the vats by machinery. No

portion of them will run out by gravity unless they contain a high percentage of water.

Since the tailings will be discharged containing small percentages of free acid and copper sulphate, the kind of metal used for disks, plows or buckets in the excavating machinery was given consideration, although it is improbable that the quantity of acid or copper sulphate in the tailings will be sufficient to prohibit the use of cast iron, steel, or perhaps a high-silicon iron which has high acid-resisting properties.

Krupp bronze and "Monel" metal were both submitted to the following tests. Pieces of each metal were allowed to stand in acid solutions and in acidified copper sulphate solutions of varying strengths—the bronze for two and one-half and the Monel metal for three months.

The bronze precipitated the copper completely from all but the two strongest solutions. The Monel metal precipitated no copper, but passed into solution to a greater extent than the bronze. The results were as follows:

With Sulphuric Acid

Strength of Acid Per Cent. H_2SO_4	Per Cent. Loss in Weight	
	Bronze	Monel Metal
0.12	3.22	3.89
0.25	3.26	5.92
0.50	3.77	8.86
0.75	3.54	7.62
1.00	3.64	7.55
1.50	4.55	10.07

With Acidified Copper Sulphate Solution

Strength of Solution, Per Cent. Cu	Per Cent. Loss in Weight	
	Bronze	Monel Metal
0.05	1.22	14.50
0.10	1.22	7.20
0.20	0.76	4.73
0.30	0.82	5.08
0.40	0.10	2.17
0.50	0.22	7.87

The Monel metal was also allowed to stand about six weeks in the following solutions:

	Loss in Weight, Per Cent.
10 per cent. sulphuric acid	2.81
10 per cent. sulphuric acid and 10 per cent. copper sulphate (2.5 per cent. Cu)	4.88
10 per cent. copper sulphate (2.5 per cent. Cu)	1.99

A high-silicon iron, under the trade name of "Duriron," was tested in a similar manner by these solutions. So far as could be observed at

the end of a few weeks this metal was not corroded in the least but the experimental work was closed before this test was finished. The acid-resisting quality of this alloy is well known but it has the disadvantage of being very hard and brittle.

Precipitation of the Copper from Solution

I have purposely divided my subject into two parts because there have been almost as many methods proposed for precipitation of the copper from solution as there have for leaching it from the ore, and to-day there is probably more uncertainty and more difference of opinion concerning the methods of precipitation than there is concerning the methods of leaching. Of all the methods proposed there is just one that has been demonstrated a commercial success, and that is precipitation on metallic iron.

Lime or limestone was used in some of the earlier methods of hydrometallurgy, but this gave a precipitate that was little better than the ore itself.

Sulphurous acid came into prominence a few years ago, but it involved difficulties of manipulation that have not been overcome, even in small-scale operations. Hydrogen sulphide, produced from iron matte, has been tried but has not been developed on a large scale and is likely to prove a troublesome reagent under such conditions.

Electrolytic precipitation is always attractive and has probably received more attention and intelligent experimentation from skilled metallurgists than any other method While some small plants use this method from time to time, it cannot as yet be called a commercial success.

When working on a metallurgical problem, where the margin is narrow, one must always consider the materials at hand as the cheapest obtainable and most likely to permit profitable operations.

As possible precipitants for the copper, I considered the following in the order named: Natural sulphides of iron, artificial sulphide of iron, sponge iron, pig iron, and electric current.

Natural Sulphides of Iron

If acid leaching is used for the oxidized ore some other process will have to be used for the sulphide ore. It is probable that this would be some form of mechanical concentration.

Tests had already been made by the ordinary methods of wet concentration at the University of Arizona before my arrival. These, by very careful manipulation, gave a recovery of 68 per cent. of the copper, 77 per cent. of the gold and 63 per cent. of the silver, but the concentrates carried only 10.2 per cent. copper from an ore that assayed 2.3 per cent. copper, 0.02 oz. gold and 0.22 oz. silver per ton.

By a combination of flotation and concentration I was able to raise the recovery to over 76 per cent. and the grade of the concentrates to nearly 14 per cent. copper, although the flotation concentrates themselves assayed nearly 24 per cent. copper. As is often the case when chalcopyrite has its source in igneous rocks, it is intimately associated with magnetite. Even fine crushing to 40 mesh failed to separate the magnetite from the chalcopyrite in this ore and this destroyed the properties of the latter for flotation methods to a great extent, as well as lowered the grade of concentrates. However, by treating these concentrates on a magnetic separator I was able to raise the grade to 18.5 per cent. copper. The magnetite itself carried no value.

This, then, was the product, and probably the best that can be obtained from the sulphide ores by the usual methods of treatment. It must be shipped to the company's smeltery at Douglas for treatment, or it must be smelted into matte and blister copper on the ground. If shipped to the smeltery, it would certainly be advisable to raise its grade if possible either with or without removing the magnetite.

It is generally conceded among geologists that secondary chalcocite is formed by the action of copper sulphate solutions on pyrite. Barring the recognized but unknown intermediate reactions, the final result is supposed to be represented by this reaction:

$$5FeS_2 + 14CuSO_4 + 12H_2O = 7Cu_2S + 5FeSO_4 + 12H_2SO_4$$

While later investigators have found this reaction "incompatible with the actual volume relations observable," there seems to be no doubt about ferrous sulphate and sulphuric acid being formed when the chalcocite is deposited, and, if the above reaction is true, nearly all of the acid originally combined with the copper is regenerated.

The reaction with chalcopyrite might be very simple, producing cupric instead of cuprous sulphide, thus:

$$CuFeS_2 + CuSO_4 = 2CuS + FeSO_4$$

I had therefore a recognized precipitant for copper in the concentrates from the sulphide ores. Its efficiency as a precipitant was important but not essential, since any precipitation of the copper from solution and consequent enrichment of the concentrates would be a distinct gain. The only factor to make it a commercial success was the speed of the reaction and in this it failed. However, having secured sufficiently speedy reactions from other natural sulphides of iron on previous occasions, I am not prepared to say the last word has been spoken in this case.

Artificial Sulphide of Iron

Two sources of this precipitant would be available. First, the matte produced by smelting the concentrates at the mine, and second, the matte

that could be produced by smelting the low-grade pyrite at Bisbee. If soluble sulphides precipitate cupric sulphide from copper solutions and natural sulphides of iron precipitate cuprous sulphide, it is reasonable to expect the artificial sulphide of iron to do the same.

Earlier experiments had shown me that iron sulphide precipitates metallic copper and not sulphide of copper from solution. This investigation was carried further and I found the reaction to be, not a simple transposition of the metals, but the following:

$$4FeS + CuSO_4 = Cu + FeSO_4 + Fe_3S_4$$

I found also that the precipitation of copper stops at this point, although free sulphuric acid will readily decompose the Fe_3S_4 with the liberation of hydrogen sulphide. Therefore less than 25 per cent. of the iron in pure iron sulphide is available for the precipitation of copper and the resulting product cannot carry more than 17.7 per cent. copper.

Increase in the copper contents of a matte decreases its precipitating power so much that a matte containing over 30 per cent. copper precipitates practically no copper from solution. Nevertheless the iron from the matte passes into solution to an appreciable extent. The only explanation I can offer for this phenomenon is a reduction of the cupric sulphate to cuprous sulphate and an oxidation of the cuprous sulphide of the matte to cupric sulphide, according to the following reaction, but I am not prepared to say that this is correct.

$$FeSCu_2S + 2CuSO_4 = Cu_2SO_4 + FeSO_4 + 2CuS$$

I could produce no condition that would make a copper matte precipitate copper from solution within a reasonable length of time.

This method of precipitation was pronounced a failure. For further details see my original paper on this subject.[2]

Sponge Iron

This was first used as a precipitant for copper in England in 1837, although Gossage, in 1859, was the first to use it in connection with the wet extraction of copper from ores. It was made by heating a mixture of finely crushed iron ore and coal in a reverberating furnace, with a reducing flame.

Since then a great deal of sponge iron has been produced by iron manufacturers and a number of furnaces have been invented for its production, all of which are described in books on the metallurgy of iron and steel. It is now produced commercially at Hoganas, Sweden, by heating a mixture of fine magnetic iron concentrates and coal in pots or retorts by means of producer gas.

[2] *Engineering and Mining Journal*, vol. xcvii, No. 15, pp. 745 to 748 (April 11, 1914).

According to Sir I. Lowthian Bell,[3] oxide of iron is easily reduced by solid carbon, but carbon monoxide gas is greatly preferred as a reducing agent.

The reduction of Fe_2O_3 begins at 420° F.

The reduction of FeO begins at 1,300° F. and is complete at 1,475° F.

Sponge iron readily combines with oxygen when exposed to the air at a red heat It is also capable, at certain temperatures, of splitting up the carbon dioxide formed by its own reduction.

The necessary reaction for the reduction of iron oxide by carbon monoxide seems to be

$$Fe_2O_3 + 9CO = 2Fe + 3CO_2 + 6CO$$

or, that there is as much oxygen in the CO escaping as in the CO_2 formed. This reaction is not so simple as it appears, however, for carbon monoxide gas is split into carbon dioxide and carbon in the presence of iron oxide or metallic iron, thus:

$$2CO = CO_2 + C$$

Hydrogen greatly assists in the reduction of iron oxide.

Dr. Frankfurter of the University of Minnesota found that finely pulverized iron ore begins to reduce at about 300° F. in an atmosphere of hydrocarbon gas and is completely reduced below 1,000° F.

S. H. Stupakoff, in a paper read before the Engineering Society of Western Pennsylvania, states that carbon monoxide begins to reduce precipitated iron oxide at 285° F., roasted carbonate at 390° F., and is active on all ores at 750° F. It is most active at 1,000° F.

Solid carbon begins to reduce iron oxide at 800° F.

The reverse action, of CO_2 being reduced to CO by metallic iron, begins at 800° to 1,100° F., depending upon mass action, and is most active at 1,475° F.

A mixture of $3CO_2$ and 2CO is oxidizing at 1,000° F. and a mixture of $1CO_2$ and 2CO is oxidizing anywhere above 1,500° F.

My plan for the production of this precipitant was to manufacture sulphuric acid at the leaching plant from the low-grade Bisbee pyrite and then reduce the iron oxide in the calcines to sponge iron for precipitating the copper. The Bisbee pyrite contains about 38 per cent. iron and 1.5 to 2 per cent. copper. The copper would pay for the mining.

The freight rate on this material from Bisbee to Ajo would be very low and divided equally between the acid and precipitating departments. The acid plant would be constructed so as to have the same life as the oxidized ore in the mine. In this way the cost of acid made at the leaching plant should compare favorably with that made at the smelting plant at

[3] *Principles of the Manufacture of Iron and Steel.*

Douglas, when the cost and maintenance of acid trains and storage equipment are considered in conjunction with the high freight rates on this commodity.

The calcines from this pyrite would be in ideal condition for the production of sponge iron. They would contain a finely divided, porous, artificial oxide of iron which could easily be reduced at low temperatures. The product obtained from the reduction of these calcines would contain 60 to 65 per cent. metallic iron in a finely divided, porous condition that would present the greatest possible surface for the rapid precipitation of copper and the copper contained in the original pyrite would all be recovered.

Laboratory experiments were made by mixing iron ore with 30 to 35 per cent. coal, both crushed to 16 mesh, and heating in a closed clay crucible in an assay muffle for 1 hr. at a temperature of 1,600° to 1,800° F. From 75 to 77 per cent. of the iron in the ore was reduced to metallic iron.

These experiments were repeated by mixing calcines with 25 to 30 per cent. coal and heating in the same manner; 98 per cent. of the available iron was metallized.

Calcines were then placed in an iron tube and heated to a low red heat in an atmosphere of hydrocarbon gas for 1 hr. Over 90 per cent. of the available iron was metallized.

This experiment was repeated and the tube was heated to incipient redness for 90 min.; 93 per cent. of the available iron was metallized. The gas used for heating the tube in this experiment was the impoverished gas coming from the reduction of the calcines.

The available iron in these experiments was the iron existing as oxide and not that existing as sulphide.

These experiments were again repeated on calcines from Bisbee pyrite crushed to ½-in. mesh instead of 16 mesh. At the end of 1 hr. 76.6 per cent. of the available iron was metallized and at the end of 2 hr. 88.3 per cent. was metallized.

Sponge iron has been produced in quantity by iron manufacturers in small reverberatory furnaces and, as I have stated, is now produced commercially in Sweden from magnetite concentrates in intermittent furnaces of the brick or pottery kiln type, so there is no doubt about the metallurgical conditions necessary for its production, nor the feasibility of its production for the precipitation of copper.

The idea of using calcines as a source of iron is new, so far as I know, and was first proposed by me during the course of these experiments in the summer of 1912. The advantages of using this material are at once apparent from every view point.

My efforts, therefore, were not to determine if sponge iron can be produced commercially, but to devise a furnace that will receive the calcines hot from the roasting furnaces, reduce them continuously and

deliver the metallized product or sponge iron to the precipitating department of the leaching plant. The problem is simply one of mechanical construction with no impossible or prohibitive features. The production of sponge iron is surprisingly easy, when the conditions are right.

After trying several small furnaces of different designs, I erected a Wedge double-function roasting furnace of special design to meet my conditions. This furnace was 9 ft. 9 in. in diameter, had six roasting

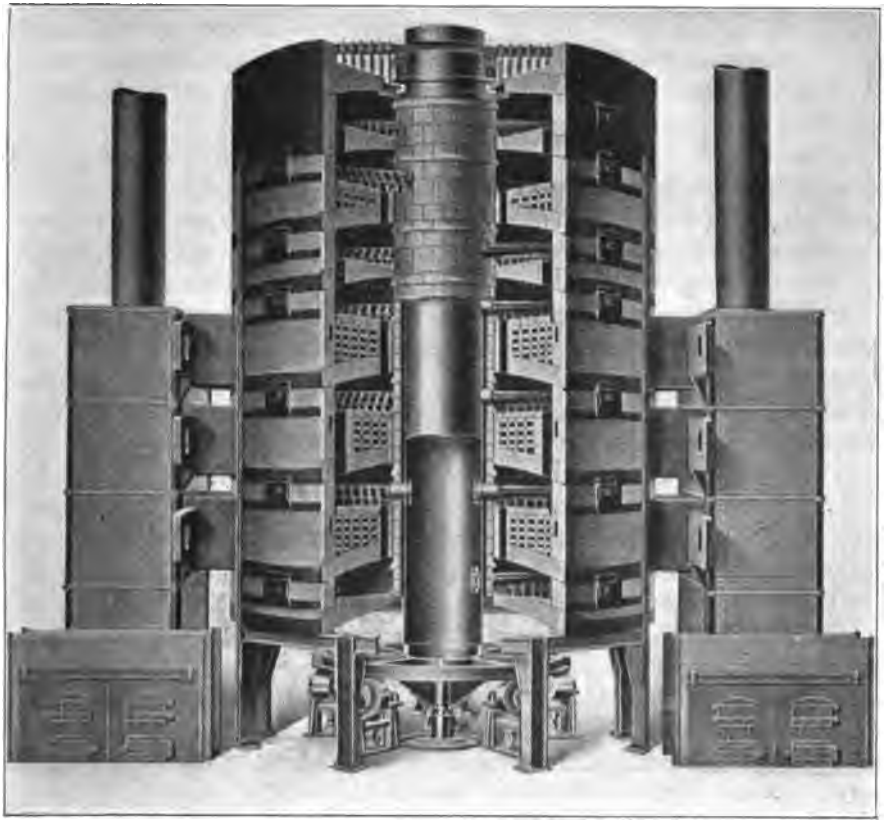


FIG. 4.—WEDGE DOUBLE-FUNCTION ROASTING FURNACE.

hearths, one drying hearth and one cooling hearth. The three upper hearths were of the ordinary type for roasting sulphide ore. The three lower were muffle hearths heated with oil from outside fire boxes. The two sets of hearths were sealed from each other by automatic cast-iron valves. The entire furnace was sheathed with sheet iron except the top and bottom. Luting rings were placed around the center shaft, and every effort was made to make the furnace as air tight as possible. Each hearth was connected with the flue so the gases could be controlled as desired.

A screw feeder was attached to the side of the furnace to deliver pulverized coal on the upper muffle hearth.

A general but incomplete idea of the furnace may be obtained from Fig. 4.

I expected to be able to roast Bisbee pyrite on the three upper hearths and deliver the red-hot calcines on the top muffle through the automatic valves, where they would be mixed with powdered coal from the feeder. In this way no heat would be lost and reduction would begin at once.

I soon found that I had made a serious mistake in trying to put two furnaces into one, for I had neither a roasting nor a metallizing furnace. With all six hearths for either purpose, I would have obtained better results.

The spaces for the passage of hot gases under the muffles were too small. In order to maintain the temperature that was desired on top of the muffle, the heat directly opposite the fire boxes was high enough to soften the clay tiles and cause them to warp. This allowed the calcines, already reduced to ferrous oxide, to drop into the joints and form a fusible slag which soon made a hole through the muffle and admitted air and products of combustion just where I wanted a reducing atmosphere.

My greatest trouble probably came from air leakage through the bottom or cooling hearth and around the central shaft. The bottom hearth was made of brick arched in the usual manner. It was plastered on the under side with cement, but with the constant contraction and expansion of the furnace it was impossible to make it air tight. I would frequently get metallization well started on the middle muffle hearth and the iron would oxidize again before discharging.

The luting rings around the central shaft, which were supposed to seal by dipping into the ore on the hearth, were not satisfactory for this purpose.

I endeavored to overcome these difficulties by producing a gas pressure inside the furnace from the combustion of the coal. In this I was partly successful, and for short periods of time, until something broke or temperatures fell, I got encouraging results.

The highest temperatures I was able to obtain on the muffles were as follows:

	Degrees Fahrenheit
Top muffle	1,000
Second muffle	1,320
Third muffle	1,460
Cooling hearth	1,040

but these were maintained only a short time before a muffle gave way.

When reducing with coal, I should have had higher temperatures or at least 1,400° on all the muffles. The soft coal used I thought

had a tendency to deposit soot on the calcines at low temperatures, which retarded the reducing action. I tried lignite coal, but with so many other adverse conditions to contend with, I could not notice any difference. I also tried powdered coke, but at that time other conditions were unfavorable and the test was not completed.

I was anxious to use producer gas, but had no opportunity to do so. My own laboratory experiments and those of all other investigators have proved that iron oxide can be reduced much more easily with gas than with solid carbon, and producer gas made from lignite or oil (which would be the cheapest fuels obtainable at Ajo) would contain the most active reducing agents known for this purpose.

I regret that my work in this direction was stopped before it was carried to a conclusion, but enough was learned to satisfy me that it will be entirely feasible to metallize calcines for copper precipitation at a cost of not over $5 per ton of metallic iron. Our estimated costs at Ajo were not much more than this, and they included half the cost of the pyrite.

I believe a muffle type of furnace can be made practicable for metallizing calcines continuously, but it must be made gas tight. The top and sides should be sheathed with iron and the calcines fed through a double bell as on an iron blast furnace or through a screw feed that would always be kept full. The bottom of the furnace should be made of cast iron or, preferably, water-jacketed segments. The central shaft should be water-sealed top and bottom.

Other material than the 3-in. thick clay tiles used in the Wedge furnace could no doubt be found for the floors of the muffles. It should be some basic material. If producer gas is used for reduction, it might be possible to make these floors of cast-iron sections, because the temperature of reduction is much lower with gas than with solid carbon.

Producer gas should be used for reduction and the impoverished gas used for heating the furnace. The Mond Nickel Co. has been reducing nickel oxide by means of carbon monoxide for years in a similar type of furnace, without accident or difficulty of manipulation.

Subsequent experiments lead me to believe that a shaft furnace can be developed that will also do the work with much less complicated operation.

In any case the metallized product must be cooled in air-tight hoppers that will seal the furnace, or discharge from the furnace directly into water. The fine iron oxidizes immediately when exposed to the air at a red heat, but if cooled in a reducing atmosphere or in absence of air and kept dry, it will remain unaltered for months. If it is discharged from the furnace into water, it must be kept under water until used. It is difficult to take it from the water and dry it without some oxidation, but it will not oxidize under water within any reasonable time.

Pig Iron

When iron is mentioned for copper precipitation, no one in this country seems to consider anything but scrap iron as available for this purpose. It takes but a small amount of investigation, however, to find that pig iron is not prohibitive in cost at almost any point in the United States not remote from the railroad. It is reported that pig iron can be delivered on the Pacific coast from China or India at a cost of $10 to $12 per ton.

The use of ordinary pig iron requires long launders and more or less handling of the iron to effect complete precipitation of the copper. Of course the rapidity of precipitation depends upon the surface exposed, so I granulated the pig iron I used. This can be done best by shattering a small stream of molten iron with a jet of steam, and then cooling in a stream of water. The product was very hard and compact and most of it was in the form of shot or pear-shaped drops. It contained 93.5 per cent. iron. A screen analysis gave the following results:

Mesh	Individual Percentages	Cumulative Percentages
+4	2.0	100.0
+8	20.0	98.0
+16	31.0	78.0
+30	28.6	47.0
+60	12.4	18.4
−60	6.0	6.0

The relative precipitating values of the different sizes of this product were obtained by treating equal weights of each size with equal quantities of pure copper sulphate solution and also with lixivium obtained from leaching the oxidized ore. Each test covered the same period of time, at the same temperature, and was made under the same conditions.

Mesh	Copper Sulphate Solution		Lixivium	
	Grams Cu Precipitated	Relative Speed of Precipitation	Grams Cu Precipitated	Relative Speed of Precipitation
+4	0.050	1.0	0.06	1.0
+8	0.105	2.1	0.11	1.8
+16	0.385	7.7	....	....
+30	0.540	10.8	0.59	9.8
+60	0.925	18.5	1.05	17.5
−60	1.175	23.5	1.31	21.8

At the Gumeshevesky mine in Russia 12 tons of granulated iron is reported to have the same precipitating capacity as 120 tons of iron plates.

The method of using granulated iron as a precipitant was also a

small problem. Sponge iron from calcines or even from iron ore is attacked inside and out by the copper solutions and each particle, large or small, is soon a mass of cement copper. Granulated iron is attacked only on the surface and unless agitated continually during precipitation it soon cements together into a solid mass which retards further action.

To overcome this difficulty, I made a tube mill out of an iron pipe 10 ft. long and 20 in. in diameter. It was fitted with 6-in. openings at each end for the admission and discharge of solution. It was filled to these openings with granulated iron and revolved at the rate of 12 rev. per minute.

The solution going into this tube mill was a neutralized lixivium from leaching the oxidized ores. It contained

	Per Cent.
Cu	1.64
Fe	0.34
Free acid	trace

It was passed through at different rates of speed to get the precipitating capacity of the mill. The results obtained from the solution passing out of the mill were as follows:

Rate, Gallons per Minute	4	6	7	10	12
Cu, per cent.	none	none	none	0.06	0.09
Fe, per cent.	.....	.....	1.98	1.88	1.84
Free acid, per cent.	.....	.....	trace	trace	trace
Iron (100 per cent.), pounds consumed per pound copper precipitated	.....	.....	1.00	0.97	0.97
Granulated iron, pounds consumed per pound copper precipitated	.....	.....	1.07	1.04	1.04

The copper precipitate contained 73.6 per cent. copper. In a previous experiment, it contained 86.8 per cent. copper.

By this method of precipitation the iron remained in the mill, always bright and clean, while the precipitated copper passed out with the neutral solution, from which it settled rapidly and could be removed by decantation. The operation is continuous.

As seen by these results, the consumption of iron need not exceed 1 lb. for each pound of copper precipitated, provided the solutions are neutralized before precipitation.

In a large plant these mills should be made longer and of less diameter than an ordinary tube mill. The shell should be made of copper and lined with silex, to prevent abrasion. The galvanic current set up by the copper and iron would hasten the precipitation.

Electrolytic Precipitation

As usual, before starting a series of experiments, I compiled all the data available on this subject. These were obtained not only from published works, but from private notes and correspondence.

The vital factors governing this method of precipitation are cost of power, material for anodes, and interference of other metals.

The power required for deposition from clean copper sulphate solutions is about eight or ten times that required for refining purposes, or about 1 kw. per pound of copper deposited. The most economical current density is between 10 and 11 amperes per square foot and the voltage required is about 2 volts, although with magnetite anodes it is said a current density of 15 amperes can be used. With an average cost of electric power of not more than 1c. per kilowatt the power costs for this method of precipitation need not be prohibitive.

The anode material heretofore has been confined to lead or antimonial lead. Fused magnetite is now being tried and is said to be very satisfactory, although fragile and expensive. Its merits are yet to be demonstrated on a large scale. Lead anodes are gradually consumed, forming the peroxide of lead, which can be collected, reduced and used over again if necessary.

Interference of other metals has proved the chief stumbling block in most of the attempts to use the electrolytic method for the recovery of copper. Arsenic and antimony are troublesome, but are seldom found in appreciable quantities in leachable ores. Iron is the principal source of trouble and with it may be considered manganese if present, for its influence is substantially the same as that of iron. Iron, by its alternate oxidation and reduction, consumes electrical energy without the deposition of copper. It must, therefore, be rendered innocuous by one of the following methods: Removal by precipitation, the use of diaphragms, or the use of depolarizers.

Removal by precipitation on a large scale may be discarded at once as impracticable, for any effort in this direction would enable one to precipitate the copper direct by means of the same amount of chemical reagents and at about the same expense.

The use of diaphragms has been tried repeatedly, but never successfully, on long-continued, large-scale operations. Under the conditions involved, it is improbable that they can ever be made to operate successfully, for diaphragm material that does the work required, increases the electrical resistance beyond commercial limits.

The use of a depolarizer seems to offer a way out of this difficulty. The most feasible depolarizer and perhaps the only one sufficiently cheap for this purpose is sulphur dioxide. This has some advantages and some serious disadvantages. Theoretically, sulphur dioxide used in this

manner will regenerate 3 lb. of acid for each pound of copper deposited. In experimental practice, 2 lb. has been the maximum obtained, owing to difficulties of manipulation. A certain amount of electromotive force is also generated in the direction of the current used, which reduces power consumption. These advantages, however, are more apparent than real. If much iron and alumina pass into solution from the ore, the actual acid regenerated in the electrolytic cells is relatively too small to be of any importance. On these Ajo ores, granting the best possible regeneration of acid from electrolytic cells, I estimated that 75 per cent. of the acid used will have to be made in an acid plant. In actual practice it would no doubt exceed this amount.

The mechanical difficulties in making the copper lixiviums absorb sufficient sulphur dioxide gas to produce the desired results have not been solved. Introduction of the gas into the electrolytic cells at the anode secures a very small absorption and renders the electrolytic plant uninhabitable for human beings. Acid makers of experience do not offer much hope of success for absorption of this gas by means of scrubbing towers.

Those who have tried to make sulphur dioxide act as a depolarizer on a commercial scale report these claims a fallacy.

Moreover, it has been found that when electrolytes containing sulphur dioxide are used the current density must be reduced to the neighborhood of 3 amperes per square foot in order to get a satisfactory deposition of the copper. This means an electrolytic plant installation three times the size ordinarily required.

It has been found impracticable to deposit copper from electrolytes containing less than 1 per cent. copper. The average lixivium including enough wash water and acid to maintain a standard electrolyte will average between 1.5 and 2 per cent. copper. Assuming then, for the sake of argument, that 35 to 50 per cent. of the copper can be deposited in good form by electrolysis (35 per cent. is actually the case in the only plant in operation), if relatively large quantities of iron and alumina pass into solution at each cycle of the lixiviant it will be necessary to discard the entire lixiviant in a short time or a portion of it at each cycle, in which the copper will have to be precipitated by a chemical reagent. This discarded lixivium will either be strongly acid or, if neutralized by fresh ore, will carry its full quota of copper. In any leaching operation, there will be also a gradual accumulation of wash water or weak solutions, which will have to be treated with a chemical precipitant.

The conditions governing the successful deposition of copper from solution by electrolysis have been determined quite conclusively after long and careful experimentation by men skilled in the art.

With ores that will yield lixiviums free from interfering elements, this method will no doubt prove a commercial success in spite of its expensive

installation, but with ores like these where the iron passes into solution as readily as the copper, when the raw ore is leached with acid, it is doubtful if electrolytic precipitation can be made a commercial success or be used advantageously.

A much more feasible method of overcoming this difficulty with iron would be to keep it from going into solution in the first place.[4] This might be done by roasting the ore after crushing to the size required for leaching. The expense would be small. While there might be some danger of forming insoluble ferrites of copper, the iron and alumina would be rendered relatively insoluble in acid, the ore would be made more porous for leaching, there would be no argillaceous slimes to absorb valuable constituents, the copper sulphides and cuprite would be rendered soluble and the lixiviums might be clean enough for electrolysis.

My work was finished before this was tried.

[4] See my discussion, *Engineering and Mining Journal*, vol. xcvii, No. 17, p. 871 (April 25, 1914).

The Treatment of Copper Ore by Leaching Methods

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(Salt Lake Meeting, August, 1914)

THE advance made in recent times in this branch of metallurgy is indicated by the attention the subject is receiving from important American copper-producing companies. Reference to the files of publications devoted to the mining industry discloses that some 20 American companies are actively investigating the amenability of their ore, or other material, to leaching methods, and that plants of varying capacities up to 10,000 tons per day are under construction or are projected. Several leaching works are in commercial operation.

It has been frequently pointed out that no method of leaching is universally applicable, for the reason that each ore differs in some particular from apparently similar ore found elsewhere, and also because local conditions are rarely the same even where the ore closely approximates in character that being worked at some other point. For this reason the older companies are approaching the matter in a conservative manner, experimentally determining for themselves the salient features in each instance.

In considering a leaching proposition three factors at once fix the attention: (1) disengaging the metal from mineralized forms in which it is found in nature; (2) recovering the metal in a commercial state after it has been dissolved; (3) the apparatus best adapted to carrying out the several operations.

Bringing the Metal into Solution

This feature has been found to present no serious difficulties. Numerous solvents for copper are known, and it has been repeatedly demonstrated on a commercial scale that from 80 to 90 per cent. of the metal contained in an ore can be rapidly brought into solution. It is interesting to note, however, that all of the companies above referred to have selected sulphuric acid as the basis for a lixiviant. Where oxidized ore practically free from precious metals is treated simple leaching with

sulphuric acid has been adopted in every case reported, less than an hour sufficing for the operation under favorable conditions.

When the copper to be extracted is found mineralized as a sulphide the use of sulphuric acid as lixiviant necessitates breaking up the sulphur combinations by roasting in order that the metal can be brought into soluble form. This operation, however, instead of being a drawback, presents advantages. The expense of roasting is light, for it is very effectively and cheaply carried out in modern mechanical furnaces, and the lixiviant itself is thereby obtained with which the copper is subsequently removed from associated gangue. In some cases it has been found that the addition of a little salt at certain stages in the roasting assists in bringing copper into solution and facilitates the extraction of any silver present.

The degree of comminution necessary in order to obtain a high extraction depends upon the mineralogical and physical characteristics of the ore. In the case of a sulphide requiring a preliminary roast, reduction to 16 mesh has usually been found sufficient. Where an oxidized ore is concerned, as the copper minerals for the most part lie along fracture planes in the matrix, comminution to $\frac{1}{4}$ in., or even $\frac{1}{2}$ in., often exposes the cupriferous portion sufficiently to permit satisfactory extraction.

If time is an element of importance, as in the case of small mines where it is not desirable to have considerable capital tied up in ore undergoing treatment, or where a high percentage of extraction is essential, fine grinding accompanied by agitation may sometimes be advantageous; but the benefits accruing from fine comminution disappear where large deposits of low-grade ore are handled. Naturally, the finer the grain, the more readily the acid solution attacks copper minerals, and the less time is required to bring the metal into solution. Then the question arises whether or not it is more economical to crush to $\frac{1}{4}$ in. and leach in large vats by percolation, or whether the ore should be reduced to 16 mesh, or finer, and subjected to agitation.

For instance, a certain ore when crushed to $\frac{1}{4}$ in. was found to yield 70 per cent. of its contained copper in three days by the use of weak acid solution. The same ore crushed to 16 mesh permitted 85 per cent. extraction in 4 hr., using a much stronger acid lixiviant. With a 2 per cent. ore this means a saving of 6 lb. copper per ton of ore treated, together with economy in time. Offsetting these advantages are: greater first cost of crushing and agitating machinery; higher maintenance charges on plant; and difficulty in washing the finer tailings. Almost any ore may be treated by percolation when properly prepared. It has been frequently noted by metallurgical writers that solutions will percolate more freely through a roasted ore than through one treated in its natural state.

Agitation of course brings copper more quickly into solution than percolation, and one advantage of this rapid action is that sulphuric acid

solutions manifest a selective action, attacking oxidized copper minerals before the iron and alumina content of an ore. Hence, in leaching by agitation the resultant lixivium is apt to be less contaminated by these elements than where percolation is applied. This is of importance when copper is to be subsequently removed from leach liquors by electrolysis, using cells based upon copper-refinery practice. It is not so important when depolarizers and moving electrodes are employed.

Many oxidized copper ore bodies occur in what are known as contact-metamorphic deposits, the gangue of which is largely composed of garnet and associated minerals, wollastonite, vesuvianite, epidote, etc. Analyses of such ore disclose large amounts of calcium oxide, which may lead to erroneous assumptions and possible rejection of leaching processes as unsuitable in such cases. The facts are, however, that these calcareous minerals are not attacked by weak acid lixiviants in the length of time necessary to extract the associated copper. It is not always safe to base a verdict upon the evidence of analyses alone when considering the appropriateness of this or that method of reduction.

When leaching with sulphuric acid lixiviants, and if the ore contains no sulphides or sulphates, the acid cost may constitute a preponderating proportion of the expense, and a source of cheap acid becomes a vital factor. On low-grade ore treated in the raw state the consumption of acid varies between 2 and 4 lb. per pound of copper produced, and when this reagent costs $0.014 per pound the outlay for this item alone may equal one-third of the total cost of the operation. This expense may be greatly reduced, or even wholly removed, when sulphides are available, for by roasting the latter and passing the gases containing sulphur dioxide into electrolyzing cells the amount of current required to deposit copper is reduced and an excess of acid accumulates for use in leaching new batches of ore. With the exercise of some ingenuity sulphur dioxide may be applied in this manner without delaying the operation and without causing annoyance to those working around electrolytic cells.

In leaching roasted mill tailings the consumption of acid appears to be small—about 50 lb. acid per ton of tailings handled—even with very fine material. This illustrates the benefit derived from roasting substances containing colloids, such as mill slimes, when the copper is to be subsequently extracted by lixiviation. Roasted porphyritic ore also requires very little acid, partly because aluminous colloid compounds are dehydrated in the process, and partly because such ore contains few ingredients soluble in the strength of solutions employed.

Referring again to the fouling of liquors made use of in cyclic leaching, experimental data recently published, as well as unpublished, indicate that the matter is not as serious as was formerly thought. In one instance where considerable quantities of ore containing large amounts of lime and soluble iron were treated experimentally with copper sulphate solutions,

ferrous sulphate was present in such quantities that it crystallized in the pumps and pipe lines, at times completely stopping the flow, and yet the results were considered satisfactory; that is, in a series of six tests 1,082 lb. of copper are said to have been deposited, under conditions mentioned, with an expenditure of 1.5 kw-hr. per pound of copper. All of these tests except one were made without use of a diaphragm. The experiments were carried out at the Greenawalt ore-testing plant in Denver.

The height of ore column permissible in percolation has been the subject of much investigation. Garnetiferous ore crushed to ¼ in. percolates freely, even when the coarser material is mixed with much fine. An ore which contains such a proportion of fines that solutions will not pass through it when crushed to ¼ in. and thrown into a vat, can be rendered permeable by dampening and thoroughly mixing before charging. By this operation the finer particles are made to adhere to the coarser, and do not collect in impervious layers, which is apt to occur when the material is charged in a dry condition.

It has been found in leaching an ore carrying large amounts of lime and iron that when the lixiviant becomes weak in acid, basic iron salts and gypsum separate and clog the interstices between the ore grains, causing the flow to cease. This does not take place when sufficient acid is present to retain iron salts in solution. With proper precautions an ore column of average oxidized copper ore 10 ft. or more in height can be percolated without difficulty; in one recently described ore-leaching plant the percolation vats are being built 16 ft. high.

Heating the lixiviant, of course, always renders it more active, but this is not always desirable in copper-ore leaching using sulphuric acid, because under such conditions combinations of elements other than those of copper are attacked and their bases are brought into solution. It is seldom found necessary to heat the liquors in leaching proper, the strength of acid lixiviant used being sufficient to effect solution of the copper at ordinary temperatures.

Most ores amenable to the leaching process carry such small quantities of the precious metals that they can be neglected. Where gold and silver are present in amounts sufficient to warrant their extraction, they are either converted into chlorides in the roasting furnace and afterward dissolved in brine, or other solvents, from which solutions they may be removed by any one of several well-known methods, or else they may be recovered from the residues in separate operations after the copper has been taken out by acid lixiviation.

Recovering the Metal from Leach Liquors

There have been many processes brought out for removing copper from solutions containing that metal, and some of these may find application in

special cases, but there are only two which have met with extended commercial use—precipitation by means of metallic iron, and deposition by the electric current. The first mentioned still has its advocates, but electrolysis is meeting with more and more favor as that method becomes better understood by men in the field, although the apparatus now in use is still poorly adapted to the purpose.

At the present time the practice so long in use at electrolytic copper refineries is being closely followed, but it is doubtful whether these methods of procedure are best suited to the requirements of ore leaching. The necessity for improvement becomes obvious when the conditions in an electrolytic cell employed in removing copper from an ore lixivium are closely studied. In copper refining the bath is kept fairly constant with regard to copper content and free acid present, fresh metal being taken up from the anodes as fast as that in the electrolyte is deposited upon the cathodes. In ore leaching, on the other hand, copper is being constantly removed from the electrolyte, and the latter has to be returned to the ore for a fresh supply. It is manifest, therefore, that conditions differ essentially in the two operations and the premises seem to call for modifications of the apparatus used.

In treating an ore lixivium, at the electrode where current enters the bath an acid radical (SO_4) is being constantly disengaged which decomposes water and combines with the hydrogen set free to produce H_2SO_4. This reaction is made apparent by the formation of oxygen bubbles on the anode; if these are not in evidence it is because the gas is entering into combination with some salt or element present in the electrolyte, a feature usually, though not always, detrimental to the process. Nascent oxygen produced in this way may attack the material of which the anodes are composed, wasting them, or it may oxidize ferrous salts in the electrolyte to ferric, thereby bringing into action a solvent which corrodes the copper being deposited on the cathodes. Ferric salts are then being formed at the anodes and reduced to ferrous at the cathodes, wastefully consuming electric current. Or anions and cathions liberated at their respective poles may set up counter currents working in opposition to the main current. To overcome complications introduced by formation of ferric salts attempts have been made to keep the anolyte separated from the catholyte by means of diaphragms, but this recourse has not met with much success in commercial operations.

It is therefore evident that in electrolysis of copper lixiviums there is urgent need of a cell in which oxygen forming at the anodes may either be removed as fast as produced, or rendered innocuous by chemical combination, operations to which the type of electrolytic bath found to meet the requirements of copper refiners does not readily lend itself. Ingenious forms of apparatus have been devised by means of which the liberated gases are shaken from the electrodes as fast as set free, and corrosion of

lead anodes is said to be thereby reduced to about 10c. per ton of copper deposited. This is an advance in the right direction, but in such cells impoverishment of the electrolyte at the cathode still assumes adverse proportions, and solutions have to be returned to the leaching department with copper content only slightly reduced.

In electrolysis, economy is effected by facilitating the supply of cathions to the negative electrode so as to avoid as far as possible waste of current in doing work other than that desired. If left to the magnetic attraction of the electrodes alone the movement is too slow to prevent impoverishment and its attendant evils, at least in commercial electrolysis of ore lixiviums.

Circulation of the electrolyte is the means usually taken for supplying necessary copper ions to the cathodes; the more rapid the movement, the more efficient the supply. Also, adhesion of oxygen bubbles to the anodes is lessened by a current of electrolyte impinging against them. Circulation alone, however, does not go far enough, as is apparent from the fact that only comparatively rich copper lixiviums can be treated in the common type of electric refining cells, and that the liquors must be returned to the ore-leaching department in some instances when only about one-fifth of the copper content has been removed. This method of operation calls for continuous pumping of liquors, which serves no useful purpose other than taking up a small amount of copper in one department and depositing it in another, both operations being incomplete. More of the copper than was just stated may under favorable circumstances be removed from a rich lixivium in a refiner's cell, but only by applying a low current density, which means a large plant for a given production. For example, an electrolyte may be reduced from 5 to 1½ per cent. or less, but the first cost of such a plant will be large. The desideratum is a cell which will rapidly deplete a 1½ per cent. electrolyte to about 0.3 to 0.4 per cent. with the same energy efficiency as in the former case, turning out at the same time a high-grade metal.

Experimental data on a fairly large scale have been obtained through application of the two factors—moving electrodes and depolarizing agents—which give a hopeful aspect to the solution of the problem involved. It is practical to get rapid deposition of metal of excellent quality from impure lixivium, but at the expense of power. However, considered from the commercial standpoint, results ranging from 1.2 to 2.3 kw-hr. per pound of copper deposited are encouraging. Energy efficiency must, of course, always be taken into account in addition to current efficiency when the respective merits of different cell types are considered, but first cost and speed are also commercial factors; the greater the current density, the less the size of electrolytic plant.

Deposition of metal on cathodes is affected by current density perhaps as much as by any other factor. This is the same as saying that

deposition is in a large measure dependent upon the amount of copper carried by the electrolyte, or which can be brought into contact with the cathodes in a given unit of time, for current density must be progressively reduced as the electrolyte is depleted of its copper if pulverulent precipitation is to be avoided. A firm deposit of copper may be obtained at current densities running into thousands of amperes per square foot, provided impoverishment of the liquors immediately adjacent to the cathodes is prevented. Arborescent accretions also are markedly absent when circulation is properly adjusted to current density.

There are other well-known causes which bring about disturbances during electrolysis, but if copper ions are supplied to the cathode as fast as needed, thereby focusing the current on the work desired of it, irregularities disappear. Whatever the origin of pulverulent deposition may be thought to be, the fact remains, that a firm bright deposit of copper can be obtained from an impure (1.5 per cent.) electrolyte when copper ions are properly supplied to the cathode, and provision is made for counteracting the injurious effects produced by liberated gases, whereas the same liquor in the electrolytic refining type of cell gives only a loose, dark-colored spongy mass which bridges the electrodes and causes short circuiting.

Heating the electrolyte is a questionable procedure, because it renders the objectionable components of the liquor more active in corroding the deposited copper. It has long been known that a greater quantity of copper can be deposited by the same amount of current from a neutral copper sulphate solution than from one of the same metallic content but carrying in addition free acid. The cause of this is that the acid redissolves the deposited copper. If ferric salts are present corrosion is still more marked. These reactions are furthered by elevating the temperature of the bath. Heating an electrolyte always facilitates passage of the current; but ore lixivium contains a variety of salts, and as all ions may serve the current as a means of transport, a large amount of electricity may find passage without depositing copper. A good conducting electrolyte is not necessarily economical in deposition of metal.

Apparatus

Concrete as a material for construction of vats, launders, etc., in leaching works has found application a number of times, but nothing is on record as to its durability under working conditions. As the cement binder is alkaline in its nature, and as leaching solutions are of varying acidity, it is to be expected that where the two come into contact chemical reactions must necessarily follow. In a measure the strength of the acid liquors will have an influence, as will also the salts contained in the solutions, but where much free acid is present ordinary concrete will crumble.

Concrete has manifest advantages over wooden construction, especially in hot, dry countries, and, of course, concrete vats are less apt to deteriorate than are wooden ones in case of temporary suspension of activities. To make the good qualities of concrete available the problem is to devise means for preventing penetration of acid liquors into the body of the concrete, which sooner or later must result in its destruction.

There are several ways in which this object may be accomplished on a small scale, but their adaptability to commercial purposes has yet to be demonstrated. The inner layer of the concrete may be made of some acid-resisting composition, or a concrete vat may be lined with brick which are not affected by the solutions, or after the vat is completed a non-corroding coating may be applied which sinks into the finished concrete and protects the binder. Heavy mineral oils, and paraffin applied hot, have been used in this manner. Coatings of pitch or asphaltum do not seem to answer the purpose well, for sulphuric acid solutions penetrate them through minute holes, forming gypsum, with consequent swelling, and the asphalt coating peels off as the concrete disintegrates.

Recently it has been stated that reinforced-concrete vats may be lined with mastic asphalt and satisfactory results obtained. The asphalt mixture had to be specially prepared and the work was done by workmen skilled in handling such material. It remains to be seen how this construction will endure under working conditions on an extended scale. It would seem to be subject to the defect common to all linings designed to protect corrodible material from the action of strong acids: if a leak starts anywhere the alkaline concrete binder will be attacked as a matter of course and this will only become known when the structure collapses.

Mixing heavy mineral oils with the concrete as it is made, affords a hard liquid-repelling mass which behaves well with weak acid solutions. The oil is incorporated with the cement-sand mortar before the broken rock is added, as in the L. W. Page waterproofing process. This method renders the whole mass to some extent acid-resistant in that the oil repels liquors seeking to permeate the structure. It has the advantage over the lining system that leaks are self-stopping and the lining is not pushed off leaving the concrete backing unprotected. Still, even oil concrete slowly disintegrates when strong (10 per cent.) acid liquors are permitted to act on it for weeks at a time.

Certain kinds of wood—notably the Western fir—have been found to resist acid solutions to a remarkable degree, even without lining. Storage vats holding cupriferous solutions containing 10 per cent. or more free sulphuric acid showed no signs of failure after being in use for several months.

With the help of reinforced concrete, leaching vats of large dimensions may be constructed. In a plant now being built the vats are designed to hold each 10,000 tons of ore. It is expected that two days' percolation

will suffice to render soluble 90 per cent. of the copper contained in this ore.

Costs

Extraction of copper from suitable ore by leaching methods is generally assumed to cost less than the combined expense of concentration, smelting, and refining, and experimental data tend to confirm this opinion. At the Butte-Duluth plant, which is handling in excess of 100 tons of oxidized ore daily, the copper is said to cost $0.085 per pound. This figure probably refers simply to working expenses and does not include freight East, marketing, etc. After the contemplated increase in the capacity of the plant it may be expected that cost of the metal will be materially reduced.

Experimental work carried out at Morenci, Ariz., indicated the probable cost of leaching tailings slime at $0.0743 per pound copper delivered in New York. This includes expenses after the copper leaves the mine, with due allowance for interest and depreciation.

From extended tests made at the Anaconda works, carried out in great detail upon a working scale, the conclusion was reached that copper could be produced from concentrating-mill tailings for about $0.07 per pound copper.

At the Steptoe mill, in Nevada, a small leaching annex is treating oxidized copper ore in conjunction with flue dust at a cost of $0.08 per pound copper recovered.

The actual cost of copper produced by leaching methods is, however, not of so much importance as the fact that ore may be handled in this manner which is not amenable to reduction by any other process. A siliceous oxidized copper ore carrying from 1.5 to 2.0 per cent. metal may be profitably leached under favorable conditions; there are no other metallurgical means known for commercially handling ore of that character and grade.

Leaching Copper Products at the Steptoe Works

BY W. L. AUSTIN, RIVERSIDE, CAL.

(Salt Lake Meeting, August, 1914)

AT the Steptoe metallurgical plant, where ore of the Nevada Consolidated Copper Co. is beneficiated, a small copper-leaching annex has been in operation treating flue dust from roasting-furnace dust chambers, together with a siliceous oxidized ore. This leaching plant was designed to obviate smelting low-grade siliceous flue dust, which interfered with the operation of the reverberatories by forming a blanket on the charge.

The major portion of the copper content of the flue dust accumulating in the upper chambers of the roaster flue, and at the stack, is water soluble, and also contains much free sulphuric acid and soluble sulphates. It was found that sufficient acid could be obtained by mixing this material with water to provide a lixiviant for leaching the siliceous copper ore mentioned. This was the sole source of the lixiviant used in the operation to be described.

An analysis of the flue dust showed it to be composed as follows:

	Per Cent.
Copper	3.00
SiO_2	23.41
Fe	4.08
CaO + MgO	2.72
Al_2O_3	6.07
Free H_2SO_4	19.94
SO_3 combined as sulphate	25.10
H_2O combined with the sulphates	10.67
O	2.50
Total	97.49
Copper soluble in dilute H_2SO_4	2.56

The oxidized ore, locally known as carbonate, has the following composition:

	Per Cent.
Copper	3.47
SiO_2	71.1
Fe	2.3
CaO	0.6
Al_2O_3	9.5
S	0.9
Total	87.87
Copper soluble in dilute H_2SO_4	3.23

A screen analysis of the "carbonate" ore discloses the distribution of the copper throughout the several sizings:

Mesh	Weight, Pounds	Copper, Per Cent.	Per Cent. of Total Copper
+ ½	19.5	3.86	21.8
+ 10	20.5	3.85	22.9
+ 20	11.8	4.14	14.1
+ 30	9.8	3.82	10.8
+ 40	5.0	3.50	5.1
+ 60	7.5	3.65	7.8
+ 80	4.7	3.47	4.6
+100	4.4	3.00	3.7
+150	3.0	2.45	2.1
+200	4.5	2.00	2.5
−200	9.3	1.76	4.6
Average		3.475	100.0

It is evident from these analyses that the copper in this ore is largely mineralized as silicate, for cryptocrystalline chrysocolla does not break into small particles as do the more brittle carbonates. The screen analysis shows that when crushed to ½ in. 70 per cent of the copper mineral remains on a 30-mesh screen. It is also interesting to note that the copper in the mineral crushed to this degree of fineness is readily soluble in dilute sulphuric acid. It has been repeatedly observed in other cases when treating ore in which the copper is present largely as silicate that a high percentage of the metal may be extracted with dilute sulphuric acid.

The method of operation pursued at the Steptoe plant consists in hydraulicking the piles of flue dust which have been withdrawn from the dust chambers, as an economical means of dissolving the soluble contents and of transportation to the tanks. The stream of water is conducted to vats, or impounded in dams, where the mud is allowed to settle. The clear solution resulting, which contains the soluble salts and is now the lixiviant, is drawn from these reservoirs to a wooden vat, fitted with a filter bottom. This is the ore-leaching vat proper. It is 40 ft. in di-

ameter and 10 ft. deep. It is filled with crushed "carbonate" ore to a depth of 7 ft.; the ore column through which percolation of ¼-in. material is effected is therefore about 7 ft. high.

The excess space in this vat is used to store the lixiviant, for the hydraulicking operation is intermittent, while ore leaching is continuous. Enough lixiviant is accumulated above the charge in this vat to continue leaching during the night.

If the filtrate from the ore-leaching vat (the lixivium) still contains considerable free acid, it is pumped back and passed through the ore bed a second time. Otherwise it is allowed to flow through a filter of "carbonate" ore, and thence over scrap iron placed in precipitating boxes.

About 85 per cent. of the free acid carried by the lixiviant is utilized in the ore-leaching vat, the remaining 15 per cent. being left in the lixivium flowing over the scrap iron, to prevent formation of basic iron salts and insuring clean precipitate. About 2 lb. of scrap iron are consumed per pound of precipitate produced.

The cost of the copper recovered from the flue dust and siliceous "carbonate" ore capping is approximately 8c. per pound.

The above data were supplied by C. B. Lakenan, General Manager, and R. E. H. Pomeroy, Acting Superintendent, of the Steptoe plant.

Experimental Leaching at Anaconda

BY FREDERICK LAIST AND HAROLD W. ALDRICH, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

THE object of the construction and operation of the 80-ton leaching plant was to test out the leaching of sand tailings on a large scale and, if possible, determine a definite method of operation, and the best construction for the larger unit which is now being built. It was also of importance to obtain some practical data as regards cost items. The 80-ton plant was not expected to be a commercial success. A leaching process on such low-grade material necessitates the treatment of a very large tonnage. Such a cost item as labor, for instance, is entirely out of proportion in such a small plant. The amounts of acid, salt, scrap iron, and fuel for roasting, however, are the same per ton in an 80-ton plant as in a 2,000-ton plant. These costs were definitely established. Also the percentages of extraction and grade of tailings were definitely determined for a large-sized unit.

Following will be found the description of the plant, the method of operation and the results obtained. A general view of the plant is given in Fig. 1.

The bins consisted of one coal bin with a capacity of 30 tons, one salt bin with a capacity of 20 tons, and a "mill tailings" or feed bin which held 70 tons. The coal and salt bins were placed with their floor level even with the firing floor of the roasting furnace. The feed bin, to give it greater capacity, discharged about 12 ft. below this point, to a 12-in. conveyor belt, which in turn discharged into a pug mill. This fed another conveyor belt, which discharged into the top hearth of the furnace.

The furnace was an ordinary six-hearth MacDougall roaster, 20 ft. in diameter. It was equipped with two fire boxes, on opposite sides, the flames entering on the third hearth. An induced draft was obtained with a Buffalo blower, worked as an exhaust fan. Just above the top hearth was an annular flue around the whole circumference of the furnace. Slots at intervals of about 18 in. led from this flue down into the top hearth. The slots were 2 in. wide by 6 in. long. This annular flue proved rather unsatisfactory, as it tended to fill up with flue dust and choke the slots, thus cutting down the draft. The volume of gas through

the furnace was 3,500,000 to 4,000,00 cu. ft. per 24 hr. at standard conditions of temperature and pressure. The temperature of the outgoing gases was about 180° C.

The furnace was air cooled, 15-lb. air entering the shaft through a 2½-in. pipe. The pressure in the arms themselves was 2½ lb. per square inch; 850,000 cu. ft. of cooling air at standard conditions of temperature and pressure was used for this purpose. This, we believe, was a great deal more than was necessary, but we did not want to take any chances on losing any furnace arms, before the experimenting was finished.

There were four drop holes in the wall between the second and third floors, and six between the fourth and fifth floors. Four holes seem to be ample. The discharge from the furnace to the cooler was an 8-in. pipe at the outer edge of the bottom hearth. A nearly dust-tight chute led

FIG. 1.—VIEW OF 80-TON LEACHING PLANT.

into the cooler. All dust made in the cooler was, therefore, drawn back into the furnace.

The cooler, shown in Fig. 2, was a rotating drum, built of ¼-in. boiler plate, 19 ft. long and 2 ft. in diameter. It had a slope of ¾ in. to the foot, and made 10 rev. per minute. At the lower end, cold water entered an annular chamber through a stuffing box, and from this chamber 26 iron pipes 1 in. in diameter carried the water to the upper end and discharged into a circular launder 18 in. from the end, and on the outside of the drum.

The calcine entered the cooler at 260° to 370° C., depending on the amount of sulphur in the feed, and was discharged at about 45° C. The rate of feed, of course, also affected the temperature of the dis-

charged calcine. At 80 tons of an average feed, the cooling water required to cool the calcine to 45° C. was 25,710 gal. per 24 hr. To cool the same to 37.5° C. required 34,666 gal. per 24 hr. The cooler discharged into a 6-in. screw conveyor, into which a water spray, to settle the dust, was turned. The screw conveyor discharged to a series of belt conveyors, which carried the moistened calcine to the hoppers of the distributing launder above the tanks.

The distributing launder extended from the center to the circumference of the tank, where it was supported by a small fiber wheel. It

FIG. 2.—CALCINE COOLER.

was driven from the conveyor belt, through a series of bevel gears. The distributor was an iron launder 6 in. wide by 8 in. deep, with slots in the bottom spaced 18 in. apart. There was a slide on each slot, so that any or all of them could be closed. A 6-in. screw conveyor, resting down in the launder, moved the calcine from the center of the tank, outward.

Two leaching tanks, 32 ft. in diameter and 12 ft. deep, were placed in line with the furnace. A conveyor belt ran over each one, so that by moving the distributor from one tank to another, and by means of a switch gate, either tank could be filled. The tanks were of redwood. Each had five discharge gates, 12 in. in diameter, one in the center, and

one in the center of each quarter of the tank. Suitable launders were provided underneath the tanks, for carrying away the leached tailings. Each launder had a stream of water entering through a 12-in. dirty-water main, from the concentrator. Valves were provided, so that all the water could be sent down one launder if necessary. Three-inch dirty-water pipes also led to the top of the tanks to furnish water for sluicing purposes. This was done with two 3-in. rubber fire-hose lines.

The filter bottoms were made with 2 by 4 in. pieces, spaced every 1 ft. across the tank, and 1-in. slats placed with 1-in. spaces across the tank in the other direction, and nailed to the 2 by 4 in. pieces with copper nails. Then the whole bottom was cut into sections convenient for handling, when taking the bottom up. A filter medium of two layers of coarse cocoa matting was spread over this. The bottom of the tank under the filter had a lead lining, which came up the sides 6 in. Suitable drains for the solutions were provided under the filter. Pipes for carrying concentrated acid were provided for each tank, and steam connections for heating the solutions, as they went on.

Two 10 by 10 ft. iron tanks were provided for holding the stock of concentrated acid. All connections and fittings for concentrated acid were of iron. As a safety, there was a lead plug which fitted into the discharge hole in the tank. This was operated from the top with an iron rod connected to the plug. There was a gauge on each tank, actuated by an air-tight iron float.

Two 28 by 10 ft. lead-lined redwood tanks held the stock solutions for leaching, and a third tank the copper solution. A heavy lead steam coil inside the tank was used to heat the solutions to prevent freezing in very cold weather. Drains from the tanks led to the air lifts, of which there were three, made of 5-in. lead pipe and lifting about 16 ft. The lower elbows were in a 16-ft. pit. The air lifts used 90-lb. air, but only because all of the available 15-lb. air was used for the furnace cooling. A very slight turn of the valve gave sufficient air. The discharge from the lifts was into a lead-lined box above the tops of all the tanks, leaching and solution. The launder system was such that it was possible to transfer solution from any tank to any other tank in the plant. A circulating air lift was provided for each leaching tank. It was not necessary to have a pit for these, however, as they only had to actually lift against a pressure of a few inches. They were connected to the bottom of the tank, under the filter, and discharged into the top, over the calcine. Each would handle 250 gal. or more per minute. Under the air-lift discharge box was a lead-lined heater box, into which live steam was injected to heat the solutions. Some of the launders were lead lined and others were not. In some places, the lead would creep and split at the seam. The wood launders with no lining were satisfactory for the time the plant was in operation, about six months, and are probably good for

a long time yet. We did not try to protect them with paint in any way A 28-ft. Oregon fir tank was put up as a spare acid tank. It was slow in taking up, but for an unlined tank it did very well. It had been put up and taken down twice before this, so it was not in the best condition. The iron hoops for 4 ft. from the bottom were protected with lead strips.

A small 9 by 10 ft. wooden tank, with agitator equipment, was also put up for prospective sponge-iron precipitation. It was not lead lined and gave no trouble from leaks.

In two of the large lead-lined solution tanks we added salt directly to the solution. It settled to the bottom, and in a day or so the tanks commenced to leak. After repairing, we discontinued this method of adding salt and were troubled no more. It seems that the hydrochloric acid liberated attacked the lead very materially. If a leak does occur in a lead-lined tank the wood backing does no good whatever. It is so thoroughly dried, of course, that it has no holding power for the solution.

To each of the solution tanks, there was an acid pipe connection and a fresh-water connection. A 1½-in. lead pipe with a bend of 1 ft. of its length at the bottom of the tank furnished 90-lb. air for agitation and mixing. This did not work very well. The air came to the top of the solution at the point where it entered, and did not agitate a very large proportion of the tank. We could get no swirling effect at all.

The lead lining and piping showed no appreciable corrosion in the six months of operation, excepting, as was stated before, where a large amount of salt came in contact with the lead, and where live steam impinged on the lead in the presence of acid solution. A lead pipe carrying steam will not stand up in our solutions when they contain 3 to 4 per cent. acid, but in the solutions after precipitation, which carried 1 per cent. or less, no corrosion could be observed in a month's continuous running. The lead air lifts showed no appreciable wear at the elbows, where the most corrosion would be expected. The lead pans in the bottoms of the leaching tanks were of no benefit, when the scheme whereby the acid solution was sprinkled on, instead of the tank being saturated, was abandoned. The leaching tanks leaked pretty badly, but this was due to faulty construction, rather than to any fault of the wooden tanks. In several places it was necessary to protect the hoops with lead strips. The air lifts worked very satisfactorily, handling a large amount of solution very easily, except when they started "foaming." This was something which we could not account for definitely. At times, the solution coming up the air lift gave a great deal of trouble because of this foam. The solutions averaged about 1.22 sp. gr. and were more or less viscous. Cold solutions seemed to foam much worse than the warm ones, so it seems as though the viscosity was what caused the trouble.

The precipitation launders were two lead-lined troughs, side by side,

each 75 ft. long, by 3 ft. wide by 2½ ft. deep. The copper solution entered the upper end of one launder and returned in the other to an air lift, which raised it into the copper-solution tank. It was circulated in this way until the copper content was low enough, say, 0.06 per cent., to be returned to the leaching system. The precipitating launders were equipped with false bottoms 4 in. above the floors to give room for the cement to settle and leave all of the iron free. A settling tank was provided and, when cleaning up, the end gates were removed and the cement washed or pushed out of the launders into it. With fresh, clean iron, this equipment will precipitate 8 to 10 gal. per minute from 2 per cent. copper to 0.1 per cent.

Owing to the entirely new and experimental nature of this work, it was found necessary to make a great many changes from the procedure which had been planned out in advance. This was true in the case of handling the material, the roasting, and the scheme of leaching.

As the plant was originally built, the feed to the furnace, as well as the calcine from the furnace to the leaching tanks, was to be handled in small bucket elevators. These gave a great deal of trouble. The nature of the material to be handled showed very soon that the elevators would not do the work. The feed has a good deal of fine material, though much coarser than slime, which tends to pack and build up as it drops to any surface. In the case of the feed elevator, the buckets would fill up and not discharge, and finally would have to be cleaned out. We endeavored to moisten the calcine as it came from the furnace, to settle the dust, and it acted much the same way as the feed. Unless the calcine was dampened as it came from the furnace, the elevator raised so much dust that it was almost impossible to work in the building. These elevators were then discarded and belt conveyors substituted. It was necessary to put in two belts to get the feed to the top of the furnace. A pug mill was installed between these two belts, to mix in a certain percentage of slime, which was to be roasted and leached with the sand.

As originally built the furnace was designed for oxychloride roasting and salt was added on the fourth floor. All the doors were machined and made air tight. There was a combustion chamber between the second and third floors. The flames entered this chamber from the fire boxes and then traveled down through six holes 8 in. in diameter, spaced around the floor of the combustion chamber, to the third floor, and up through the drop holes to the second and to the top and out. The third floor was sealed from the fourth. A 3-in. drop hole on each side of the shaft, near the center, formed the bottom of a small hopper, which was set down in the brick. There was an adjustable gate on this drop hole, which could be set so the hopper had sand in it all, or nearly all, of the time, thus preventing any gases from the fourth hearth being drawn above into the third hearth. A fan furnished the draft for the

upper three floors, and another forced the gases from the lower four through a scrubbing tower. This was for the purpose of separating the chloride or copper- and silver-bearing gases from the sulphur or waste gases.

There was a seventh floor provided. This was nothing but a flat water jacket to cool the calcine. There were also copper rakes on the seventh floor, in case it was desired to use a water spray to assist in the cooling and also settle the dust.

The bottom floor discharged through a 6-in. hole into a cast-iron pipe which contained a screw conveyor. This closed conveyor carried the material from the center of the furnace to the calcine elevator. In this way no air could enter the lower or chloridizing floors, except what was allowed to go in through an open door regulated to suit conditions. Theoretically, this looked very good, but under operating conditions a great many difficulties arose. The use of the combustion chamber required much more fuel than direct firing into the third hearth. At the time the combustion chamber was taken out, the sealed drop holes between the third and the fourth floors were also discarded, and a regular center drop hole was put in. Then the upper fan was the only one used and all the gases were combined. The fuel percentage immediately dropped down to what it should have been. This was due to the greater efficiency without the combustion chamber, and also very largely to the benefit derived, on the top two floors, from the hot gases rising from the fourth, fifth, and sixth floors, where a large part of the oxidation went on. When the furnace was run with the fourth floor at 930° F., which temperature gave the best results, there was no oxidation on the second floor from the top. The sand was heated up, of course, and just about hot enough to ignite the sulphur as it dropped to the third or fired floor. The most of the oxidation took place on the fourth floor.

When the hot calcine dropped to the water-jacket floor the jacket bulged and the rakes cut through, causing it to leak. This caused a great deal of trouble, as the water-soluble copper in the calcine went into solution and precipitated out on the iron, taking it into solution. The water jacket was also a very inefficient cooler. It was discarded, and a discharge hole cut in the outer edge of the sixth floor.

Then the problem of an efficient calcine cooler presented itself. A water jacket 20 ft. long and 4 ft. wide was built, and connected to a Wilfley table motion. This scheme worked poorly in a mechanical way, as the weight of water in the jacket was considerable, and the jerk of the jacket caused a sort of a hydraulic-ram action inside it, which caused many leaks. Also, the cooling efficiency was not very good. A layer of the finer calcine formed on the jacket and the coarser material merely slipped down over the top of this, getting very little cooling. A coil of cooling pipes, the whole length and width of the jacket, was laid on

top of it, and calcine allowed to run down over these pipes. In this scheme the calcine acted the same way, most of the calcine never touching the jacket, but sliding down over the top of the lower calcine layer. The next method tried was the rotary cooler described in the preceding pages. It worked very satisfactorily, giving an efficient cooling and requiring very few repairs. The cooler discharged into a screw conveyor. About 1 per cent. water was sprayed on the calcine at this point to settle the dust. Mechanically the screw worked very well, but the water-soluble copper ate out the iron rapidly, and made it necessary to replace it often.

A push conveyor was built to replace this screw. After proper adjustment, it worked quite well, mixing in the water nearly as well as the screw. In comparison with this, a rotating wooden drum 2 ft. in diameter and 12 ft. long, was also tried. This gave the best results of any arrangement so far, for a short conveyor, particularly where it was necessary to mix in a small amount of water, for dust settling or cooling purposes. No iron could come in contact with the moistened calcine, and the repairs were very low. It rotated slowly and mixed the water in perfectly, about 1 per cent. water being required to prevent the calcine from dusting.

The scrubbing tower for the chloridizing gases did not work very efficiently and an electrical precipitation unit was installed. It was put in by Linn Bradley, of the Research Corporation, and was designed to handle all of the gases, chloride, sulphur, and coal gases. The treater was built after the design of Dr. F. G. Cottrell, of the Bureau of Mines, and worked satisfactorily, as 90 per cent. of the copper and a considerable amount of acid were recovered from the fumes.

After a 50-day successful run, it was decided to discontinue the feeding of salt into the furnace, for a time, and see what results could be obtained. This method offered many advantages. No valuable metals were volatilized, and no accretions formed in the furnace to cause trouble. The results from the straight oxidizing roast proved to be so good that it was continued for 70 days. The extraction on the copper was even better than on the chloridizing roast, but the silver recovery was not so good. However, the excess silver that would be saved by roasting with salt would not pay for the increased expense and bother entailed by a salt roast. The leaching solutions carry salt, and a great deal of chloridizing is done in the tanks.

The results from the oxidizing roast have this advantage over those of the chloridizing period: The oxidizing period was the later one, and naturally everything ran more smoothly and better results were more easily obtained. Also there was a leaner feed to work on during the chloridizing period.

An attempt was made to roast and leach a 10 per cent. slime tailings

mixture. The slime, dewatered on an Oliver filter, was carried by a conveyor belt to a pug mill, where it was mixed with the coarse tailings. The mixture was fairly uniform, but some pieces of slime the size of a walnut entered the furnace, did not break up, and were roasted on the outside only. The slime, even after roasting, decreases the percolation rate so much as to make its treatment by mixing with the sand impossible.

The operation of the furnace, after everything was systematized, was very simple. The firing takes a certain amount of skill and experience, and the regulation of the furnace also takes some practice. A pyrometer was used on the fourth floor, but an experienced man could regulate the furnace and keep it within 30° of the required temperature, by the appearance of the lower three floors. To do this, of course, the same grade of feed must be maintained. A marked change in the amount of sulphur in the feed necessitated a readjustment of conditions in the furnace. One point is rather important; that is, not to let the temperature get too low, say 880° F., on the fourth floor, for if it once starts down it is very hard to get it back without lowering the amount of feed. The fire caused by the burning sulphur seems to go out, and then it has to be ignited again, using a disproportionate amount of coal.

The feed averaged about 3.00 per cent. sulphur and the calcine about 0.6 per cent. sulphur.

It was our experience that the higher temperatures, say 1,000° or more, caused a large amount of ferrite or insoluble copper; that is, insoluble in any acid but hydrofluoric; 900° to 930° F. seemed to give the best results.

The feed to the furnace averaged about 5 per cent. moisture, but very often went as high as 8 or 10 per cent. When this happened, the material on the top floor would bank up over the arms, instead of going through between the rakes. This always made it necessary to shut the feed off and let the rakes clear themselves. This trouble often limited the amount of feed it was possible to put through the furnace. With three or four arms on this floor, and with a feed carrying 0.68 per cent. copper and a proportionate amount of sulphur, the furnace handled very close to 100 tons per day, dry weight. The rakes on the first or top hearth were the only ones which showed any appreciable wear during the six months' operation. They were replaced about every four or five weeks. The blades and blade holders should be used there, as replacing these is much cheaper than replacing the whole rake.

Fire brick was used in the fire boxes and in the construction of the second and third floors. All the rest of the lining and the floors were built of ordinary red brick. No special endeavor was made to increase the water-soluble copper. It seemed to affect the acid consumption very little and electrolytic precipitation is out of the question at present,

since the solutions carry chlorides. About 40 per cent. of the copper was soluble in water.

To give a full idea of the experimenting and changing in the leaching schemes, it seems best to take them up separately and show what was done on each.

In the experimental work done in the summer and fall of 1912, in a small experimental plant, a leaching scheme was worked out which seemed to give excellent results. This method consisted of sprinkling the No. 1 or weak solution on the charge, then letting it stand a few hours and following with the No. 2 or strong solution in three portions. This was sprinkled on in the same way, much as a lawn would be sprinkled, except that a few hours were allowed to elapse before the addition of each portion. The solution was allowed to drain out as fast as it would and no hydrostatic pressure built up in the tank. The No. 1 solution was precipitated, made up in acid, and used as the No. 2 solution for the next leach. The No. 2 solution was returned and used for the No. 1 in the next leach.

Accordingly, an automatic sprinkler was designed for this plant, and a lead pan was put into the bottom of the leaching tank, to prevent leakage.

The first six leaches were all subjected to the sprinkling system, and were treated in much the same way, except variations in acid and salt strengths. These show in the table of tank leaches. As far as the "sprinkling system" goes, it seems to have done about as good work as the later "flooding system," considering the class of calcine used at that time. The furnace was not making very good calcine until Oct. 20, and the first four leaches had been completed at that time. However, the flooding system offers fewer difficulties and dispenses with all automatic sprinklers, and, if anything, does better work. When a tank is flooded with solution, all parts of the calcine are apparently more apt to be saturated with acid than by the sprinkler system. However, the average results of the first six leaches come as near to the results from the laboratory leaches as those of the later ones.

Leach No. 7, the first one in which the flooding system was used, gave rather poor results. This was due entirely to improper washing. The tailings sample after washing in the laboratory gave a result of 0.096 per cent. copper in comparison with 0.135 per cent. copper after the wash in the leaching tanks. Until this leach, at times there was a good deal of fine material and a good deal of slime in the feed. This hindered the percolation, and resulted in poor extraction and imperfect washing. Classifiers were installed in the mill to give a more perfectly deslimed feed. Leach No. 8 was very similar to leach No. 7. However, it gave a considerably better extraction percentage on the copper, but

this was probably due to the fact that the calcine was richer. This leach did not get a perfect wash.

In leach No. 9, a new scheme was tried. Air under 90 lb. pressure was connected under the filter bottom of the tank. The No. 1 solution was added, circulated down to 0.5 per cent. H_2SO_4, and replaced with the No. 2 solution. This was then drained for about 8 hr. In that time, most of the solution which would drain had come off. The air was then turned on for 8 hr. This was repeated three times, the air coming up through the warm, moist calcine being about 40° C. and it seemed as though its oxidizing action on the calcine was sure to benefit the extraction. This was tried in some of the later leaches, but seemed to help them very little, if at all. Leach No. 9 gave an exceptionally good extraction. However, it is doubtful, from later results, whether the aeration was responsible, as the calcine was very good and also the percolation. In this leach, the top and bottom of the tank were sampled separately. The bottom showed about 1 per cent. poorer extraction than the top.

Leach No. 10 was made on a 6-ft. depth, or 224 tons, of calcine. An attempt was made to determine if the shallower bed would equalize the results in the top and bottom of the tank. The extraction on the bottom was 2 per cent. poorer than on the top. On leach No. 11, a copper and silver balance was made, in which the calcine, tailings, and all solutions were measured, sampled, and assayed. This leach was the first of the oxidizing-roasted material. It had the same treatment as leach No. 10, except that one portion of the No. 2 solution was added under the filter bottom and forced up through the calcine. This method gave the bottom of the tank the benefit of the first application of the warm strong acid. The results show a much better extraction at the bottom than at the top of the tank. There is such a wide variation in the other direction, that it looks as though perhaps there was an error in sampling or analysis. On none of the succeeding leaches where this was tried was the extraction better at the bottom than at the top of the tank. This method is worth consideration, at any rate.

Leach No. 12 had two portions of acid added at the bottom and two portions at the top. In this case, the top gave an extraction nearly 3 per cent. better than the bottom, in spite of adding the solution at the bottom. Aeration also was used on this tank and the air heated before going in. Leach No. 13 had no aeration and all of the solution was added at the top. Leach No. 14 was started with an H_2SO_4 solution with no salt. It was planned to run this solution through, and determine whether it would dissolve enough of the copper so that it could be precipitated on the scrap iron and all of it be discarded. In this way, the chloride solutions would not have so much copper to take up, and, therefore, when

precipitated, would not take so much iron in solution or become so foul. This scheme would cut down the amount of discard solution and salt loss. The first soultion only dissolved about one-half of the copper and the cycle was prolonged to such an extent that it seemed advisable to abandon the idea. A balance was also made on this leach and 98.5 per cent. of the copper was accounted for. In this leach the silver extraction was poor. An attempt was made to put the wash water on at the bottom; it was then changed about and started on at the top. In this way, some silver which was held in solution by the salt was precipitated in the tailings, due to the dilution with the wash water. The silver results on this leach were thrown out when averaging the results.

Leach No. 15 gave the best results of all the leaches. It had rather a high calcine—0.713 per cent. Cu—and percolated very readily and had about a six-day treatment. In this case, the strong acid necessary to bring the solutions up to strength was added to the stream of solution as it went on to the charge; 55 lb. of acid per ton of calcine was added in this leach. The No. 2 solution was put on and drained, then another portion put on to re-saturate it, and to this portion enough acid was added so that a zone of 15 per cent. acid traveled down through the charge. The repeated draining and re-saturating seem to help considerably in the extraction. A new scheme was tried in the washing which seemed to give excellent results, and was continued through all the rest of the leaches. Salt, to the amount of 1 per cent. of the weight of the charge in the tank, was spread on top of the leached calcine just before the wash water was added. This strong zone of salt solution traveled down through the charge and dissolved or washed out any silver chloride remaining. The salt going down first, went into the solution tanks to take the place of the discard solutions; thus no, or very little, salt found its way into the wash-water tanks.

Leach No. 16 was a very successful one, the percolation being very good. About 50 lb. of acid was used per ton of calcine, and added to the No. 2 solution as it went on. Aeration was used. The No. 1 solution was added and circulated down to 0.6 per cent. H_2SO_4 from 4.2 per cent. Then it was replaced with one-half of the No. 2 solution. This first half was then drained, and followed down with the second half of No. 2 solution, this portion having part of the strong acid added to it. The tank was then drained to about 10 per cent. moisture and aerated for 8 hr. The air coming up through the calcine, moistened with, say, an 8 to 10 per cent. H_2SO_4 solution, was expected to give excellent conditions for oxidation and subsequent solution. The air may be, and probably is, of some advantage, but it is so small as to be almost inappreciable. Also, aeration tends to prolong the cycle considerably. The repeated draining and re-saturation are thought to be more beneficial than the aeration. When a charge is drained and then re-saturated, all particles are bound to get a fresh

supply of acid, while circulation does not accomplish quite the same end. During circulation, the solution travels slowly down through the charge, taking the easiest course, and if there are any softer portions they get more than their share of the acid, and harder places get less. In this leach there was about 1 per cent. difference in extraction between the regular extraction and that using a laboratory washed tailings as a basis for calculation.

Leach No. 17 was treated the same as No. 16, with the exception that the whole amount of acid was added to the second half of the No. 2 solution as it went on. This brought the acid strength up to about 18 per cent. H_2SO_4 for this portion. This was allowed to travel down through the calcine, replacing the first half. The charge was then aerated and the first half of the No. 2 solution returned, and the total No. 2 solution circulated was down to the same acid value as the No. 1 solution had when it went on. There was rather a large difference between the extraction on the top and bottom, and the washing could have been improved upon also.

In leach No. 18, the same solutions which were used in No. 17 were used without precipitating out the copper. They were made up in acid strength, however. The idea was to see if it was possible to run the solutions higher in copper content, without interfering with efficient leaching. The results were not very satisfactory. Using the richer copper solutions made washing of the charge much more difficult and also the variation between extractions on the top and bottom was larger than it should have been.

Leaches 19 and 20 were run side by side and at the same time. Both were treated with No. 1 solution, and then one had No. 2 solution added to it, and its drainage of No. 2 solution was lifted to the top of the other charge. The two leaches were see-sawed back and forth in this way several times; that is, one was drained as fast as possible and the drainage put on top of the other tank, adding, of course, its proportion of strong acid. This was to accomplish the large number of drainings and re-saturations spoken of above, without the use of spare or sump tanks. The scheme worked very well as far as time and the handling of solutions went. It is possible to drain a 10-ft. depth of this material to about 10 per cent. moisture in 6 to 7 hr. Leach 19 had 49 lb. H_2SO_4 per ton added to it and 48 lb. was added to leach 20.

The sluicing of tailings is a very simple and cheap operation. Seven million gallons of water per 24 hr. will sluice and carry away all the tailings which two men can sluice out of five 12-in. gates in the bottom of a tank. This amount of water was available and used, but probably much less than this would be sufficient. The greatest difficulty is at the start, when a hole is first opened up. The sand caves in and it takes a very large amount of water to carry away this sudden influx of sand

into the tail race. Two men could sluice 400 tons from one of the tanks used here in about 5 hr.

There is not much to be said with reference to the precipitating. The launders were filled with scrap iron and the solution circulated through them. At 30° C., a 2 per cent. Cu solution, carrying 0.7 per cent. H_2SO_4, was precipitated to 0.1 per cent. Cu when circulating at the rate of 8 to 10 gal. per minute.

This strong copper solution precipitated as a very good cement copper. It was flocculent, and, on wrought iron, merely touching the iron would break the copper off and leave a fresh iron surface. The cement ran from 75 to 85 per cent. copper.

The hard or cast scrap iron was also tried. At first, the copper seems to precipitate on it as fast as on the soft iron, but a layer of copper quickly plates on it, and the cleaning up is a pretty expensive operation. Every piece must be handled separately, brushed, and most often scraped, before the iron is fit for further precipitation. By using malleable iron scrap, sweeping is hardly necessary, if the iron can be moved a little occasionally. The copper drops off and to the bottom of the launder immediately. The iron consumption was about 1.15 lb. per pound of copper.

General

A copper tube was used for all tailings sampling. Iron cannot be used because of the precipitation on it of any copper in solution. The copper tube was closed at the lower end and brought to a sharp point. A slot was cut its full length, and one side bent out a little to form a cutting edge. A piece of rubber hose was pushed into the sampler, just filling it, then it was driven down through the charge, the hose withdrawn and the tube turned several times. In this way, the cutting edge filled the tube with tailings representing that part of the tank. This sampler was checked several times, by opening up a hole to the bottom of the charge and sampling up the side of the hole with a wooden scoop. If there was any abrasion of the copper off the tube, it did not appreciably affect the sample.

It requires about 20 per cent. of the weight of the calcine in weight of solution to saturate the charge. This assumes the solutions at a specific gravity of 1.00, while in reality they have a specific gravity of about 1.25.

It requires about 40 per cent. of the weight of the charge in wash water for a reasonably good wash.

The acid consumption will not exceed 50 lb. H_2SO_4 per ton of charge and the salt lost in solution discard will not be more than 1 per cent. of the weight of calcine, probably much less. By figuring acid consumptions on each leach, and using acid content of each solution, before going on and after coming off the calcine, the acid consumption on the first

14 leaches averages about 25 lb. per ton. On the last six, 50 lb. of acid per ton was added. This may have given somewhat better results, but not very much so. It seems as though perhaps the extra acid of the 50-lb. consumption was used in taking up iron and alumina, and had very little effect on the copper remaining in the charge.

The silver has a tendency to precipitate on the lead linings to a small extent. This would not offer much difficulty, as the lead rapidly becomes coated and precipitation ceases. The solutions, on cooling and on the addition of water, drop some of the silver out as silver chloride. This forms in the bottom of the solution tank to more or less extent and has to be cleaned up periodically. All of the silver in solution precipitated in the scrap-iron launders with no trouble whatever.

Summary

The accompanying Tables I, II, and III speak for themselves, so there is not much to explain about results.

The feed from the mill averages between 0.65 and 0.70 per cent. copper, about 5 per cent. moisture, and about 3 per cent. sulphur. Ninety tons per 24 hr. of this feed can be roasted in a 20-ft. MacDougall furnace of the type used here with a coal consumption of 2.75 per cent. of its dry weight. The calcine will contain about 0.6 per cent. sulphur. About 40 per cent. of the copper is water soluble.

Screen Analysis of the Feed

Mesh	Per Cent.	Cumulative, Per Cent.
On 14	4.50	4.50
On 40	50.31	54.81
On 60	15.33	70.14
On 90	11.45	81.59
On 110	6.54	88.13
On 160	5.52	93.65
On 200	2.86	96.51
Through 200	5.31	101.82

Under conditions which would apply in a large plant, it is expected that an 85 per cent. extraction can be made on the copper, and very probably 86 or 87 per cent. A tailing carrying between 2 and 2¼ lb. of copper per ton should be made. In a plant as small as this 80-ton plant, and built for temporary use, a great many petty annoyances are encountered, as the running of the furnace, and therefore of the whole plant, depended on so many small pieces of machinery in the conveying and cooling equipment. Not 10 per cent. of the furnace shutdowns were caused by troubles with the furnace itself. Naturally, the furnace

does much better roasting, and with a less coal consumption, when it is running continuously or nearly so.

No salt is necessary in the roast for the better extraction of the copper. It would increase the silver extraction from 75 to 90 per cent., but this extra 15 per cent. of the silver recovered would not pay for increased cost of salt, installation, and operation of an apparatus to catch the values in the gases. This recovery of fume values would probably not be much over 90 per cent. The chloridizing of the silver is done in the leaching tanks.

Summary of Results from Laboratory Leaches During Oxide-Chloride Period

The extraction percentages on both copper and silver assume complete recovery of volatilised values.

No silver assays were made on the feed, an average result of 0.55 oz. per ton being used in figuring extraction.

Recovery of copper, per cent	86.7
Recovery of silver, per cent	94.3
Pounds copper per ton in feed	12.48
Pounds copper per ton in tailings	1.64
Pounds copper per ton recovered	10.84
Average feed per 24 hr., tons	90.8
Average salt consumption, per cent	1.04
Average coal consumption, per cent	2.94

Summary of Results from Tank Leaches

Oxide-Chloride Period

The extraction percentages of silver and copper during the chloridising period are figured by using the copper and silver values in the feed and assuming all volatilised values recovered.

Pounds copper in feed per ton	12.04
Pounds copper in tails per ton	2.54
Pounds copper recovered per ton	9.50
Per cent. extraction on copper	78.9
Oz. silver per ton in feed	0.55
Oz. silver per ton in tails	0.15
Per cent. extraction on silver	72.9

Oxide Period

Pounds copper in calcine per ton	13.58
Pounds copper in tails per ton	2.12
Pounds copper recovered per ton	11.46
Per cent. extraction on copper—top ½ of tank	86.2
Per cent. extraction on copper—bottom ½ of tank	82.8
Per cent. extraction on copper—total tank	84.4
Per cent. extraction copper—tails washed	86.0
Oz. silver in calcine per ton	0.57
Oz. silver in tails per ton	0.15
Per cent. extraction on silver	70.9

TABLE I.—*Oxide-Chloride Roast. Results from Laboratory Leaches*

Date		Rate of Feed Dry Weight	Per cent. Moisture in Feed	Per cent. Coal	Per cent. Salt	Per cent. Copper in Feed	Per cent. Copper in Calcine	Per cent. Volatilized Copper	Average Temperature of Roast	Per cent. Copper as CuS in Tails	Per cent. Copper as Ferrite in Tails	Per cent. Total Copper in Tails	Oz. Silver per Ton in Tails	Extraction per cent. on Copper	Extraction per cent. on Silver
Oct.	21	84.7	4.5	3.00	1.16	0.587	0.504	0.083	1,070	0.048	0.060	0.108	0.05	81.7	91.0
	22	88.1	7.0	3.29	0.70	0.580	0.539	0.041	1,050	0.044	0.061	0.105	0.03	88.0	94.6
	23	101.2	4.0	3.38	1.24	0.590	0.531	0.059	1,060	0.045	0.068	0.113	0.05	81.0	91.0
	24	103.1	6.5	2.75	1.16	0.555	0.478	0.077	1,000	0.039	0.044	0.083	0.03	85.0	94.6
	25	102.0	7.5	3.00	0.93	0.501	0.460	0.041	1,000	0.036	0.029	0.065	0.03	87.0	94.6
	26	104.7	5.0	2.81	0.86	0.478	0.448	0.030	930	0.040	0.022	0.062	0.02	87.1	96.5
	27	104.6	5.0	2.82	0.97	0.545	0.521	0.024	950	0.038	0.025	0.063	0.04	88.4	92.8
	28	103.5	6.0	2.95	0.94	0.549	0.519	0.030	1,000	0.043	0.036	0.079	0.03	85.9	94.6
	29	103.6	6.5	3.16	0.87	0.555	0.501	0.054	1,000	0.040	0.039	0.079	0.02	84.0	96.5
	30	104.6	5.0	2.68	0.86	0.555	0.466	0.089	1,000	0.044	0.046	0.090	0.03	83.8	94.6
	31	109.9	5.5	2.84	0.84	0.596	0.519	0.077	1,000	0.044	0.046	0.090	0.03	85.9	94.6
Nov.	1	108.3	5.5	2.90	0.86	0.596	0.549	0.047	980	0.044	0.044	0.088	0.03	85.4	94.6
	2	104.1	5.5	2.94	1.04	0.614	0.553	0.061	930	0.043	0.028	0.071	0.03	88.5	94.6
	3	104.7	5.0	2.92	1.03	0.550	0.516	0.034	900	0.042	0.024	0.066	0.03	82.8	94.6
	4	105.0	5.0	3.24	1.14	0.562	0.533	0.029	900	0.040	0.020	0.060	0.03	89.4	94.6
	5	104.7	5.0	2.96	1.06	0.550	0.528	0.022	900	0.050	0.029	0.079	0.05	85.6	91.0
	6	103.5	6.0	2.82	1.01	0.597	0.562	0.017	920	0.050	0.020	0.070	0.03	87.8	94.6
	10	100.3	8.5	3.03	1.16	0.699	0.643	0.056	930	0.046	0.026	0.072	0.03	89.8	94.5
	11	103.6	6.0	2.92	1.15	0.712	0.661	0.051	930	0.067	0.022	0.089	0.03	87.7	94.6
	12	91.9	10.5	3.21	1.26	0.612	0.556	0.056	960	0.038	0.038	0.076	0.02	87.7	96.6
	13	91.3	9.3	3.13	1.37	0.625	0.556	0.069	920	0.042	0.033	0.075	0.04	88.1	92.8
	14	91.5	6.0	2.72	1.25	0.587	0.562	0.025	930	0.043	0.032	0.075	0.04	87.2	92.8
	15	93.7	6.0	2.93	1.25	0.656	0.619	0.037	930	0.065	0.027	0.092	0.03	86.0	94.6
	16	96.0	4.0	3.03	1.10	0.675	0.662	0.013	1,000	0.076	0.029	0.105	0.05	84.5	91.0
	17	86.5	6.0	3.50	1.22	0.644	0.619	0.025	1,020	0.060	0.045	0.105	0.03	83.7	94.6
	18	83.4	4.0	2.74	1.25	0.669	0.637	0.032	950	0.045	0.037	0.082	0.03	87.8	94.6
	19	80.1	6.0	2.80	1.22	0.700	0.650	0.050	940	0.064	0.022	0.086	0.02	87.7	96.5
	20	80.8	5.0	2.79	1.19	0.710	0.704	0.006	930	0.078	0.024	0.102	0.04	85.7	92.8
	21	80.2	5.0	2.99	1.12	0.761	0.742	0.019	1,000	0.072	0.034	0.106	0.05	86.0	91.0
	22	78.9	5.5	2.97	1.18	0.692	0.616	0.076	960	0.054	0.033	0.087	0.04	87.6	92.8
	23	77.0	4.0	2.88	1.10	0.660	0.597	0.063	960	0.053	0.025	0.078	0.02	88.2	96.5
	24	76.2	5.0	3.08	1.19	0.692	0.660	0.032	960	0.086	0.015	0.101	0.03	85.6	94.6
	25	75.2	6.0	2.75	1.24	0.666	0.609	0.057	1,000	0.048	0.017	0.065	0.03	90.4	94.6
	26	75.2	6.0	3.04	1.10	0.622	0.560	0.062	1,000	0.042	0.023	0.065	0.02	89.6	96.5
	27	79.5	4.5	2.58	0.92	0.684	0.610	0.074	1,010	0.055	0.022	0.077	0.02	88.9	96.5
	28	78.7	6.0	2.91	0.97	0.912	0.874	0.038	1,020	0.076	0.031	0.107	0.03	88.4	94.6
Dec.	2	81.4	3.5	2.60	0.83	0.595	0.533	0.062	980	0.054	0.021	0.075	0.03	87.5	94.6
	3	82.1	3.0	2.62	0.88	0.651	0.620	0.031	1,020	0.048	0.018	0.066	0.04	89.8	92.8
	4	78.4	7.5	2.77	0.95	0.620	0.558	0.038	1,000	0.055	0.014	0.069	0.03	89.0	94.6
	5	82.6	3.5	2.62	0.77	0.614	0.571	0.057	1,000	0.038	0.027	0.065	0.04	89.3	92.8
	6	82.0	4.5	3.09	0.78	0.614	0.601	0.087	950	0.046	0.028	0.074	0.03	88.0	94.6
	7	82.2	4.0	3.02	0.84	0.614	0.539	0.025	950	0.047	0.029	0.076	0.03	87.8	94.6
	8	75.7	11.5	3.12	0.87	0.614	0.595	0.081	950	0.047	0.031	0.078	0.03	87.4	94.6
	9	73.6	14.0	2.75	0.90	0.645	0.558	0.013	950	0.049	0.048	0.097	0.02	85.0	96.5
Average		90.8	5.9	2.94	1.04	0.624	0.578	0.046		0.050	0.032	0.082	0.03	86.7	94.3

TABLE II.—*Oxide Roast. Results from Laboratory Leaches*

Date		Rates of Feed Dry Weight	Per cent. Moisture in Feed	Per cent. Coal	Per cent. Copper in Feed	Per cent. Copper in Calcine	Average Temperature of Roast	Per cent. Copper as Cu_2S in Tails	Per cent. Copper as Ferrite in Tails	Per cent. Total Copper in Tails	Oz. Silver per Ton in Calcine	Oz. Silver per Ton in Tails	Extraction per cent. on Copper	Extraction per cent. on Silver
Dec.	10	81.0	5.5	2.72	0.580	0.568	950	0.047	0.030	0.077	0.44	0.07	86.7	84.2
	11	81.8	4.5	3.11	0.600	0.605	950	0.044	0.019	0.063	0.45	0.07	89.4	84.5
	12	82.6	4.0	2.72	0.584	0.574	960	0.051	0.020	0.071	0.44	0.08	87.6	81.8
	13	82.7	4.0	2.80	0.642	0.660	950	0.055	0.017	0.072	0.48	0.08	89.2	83.3
	14	81.2	5.0	2.87	0.672	0.710	940	0.074	0.017	0.091	0.62	0.10	87.3	84.0
	15	77.2	5.5	3.54	0.735	0.722	950	0.078	0.015	0.093	0.59	0.10	87.0	83.2
	16	78.7	4.5	3.05	0.710	0.703	970	0.079	0.041	0.120	0.56	0.09	82.9	83.9
	17	78.6	5.5	2.69	0.641	0.641	940	0.050	0.032	0.082	0.52	0.09	88.1	82.7
	18	81.0	6.0	2.50	0.732	0.707	930	0.060	0.019	0.079	0.55	0.07	88.8	87.3
	19	78.7	7.0	2.73	0.700	0.713	930	0.073	0.018	0.091	0.53	0.08	87.0	84.9
	20	77.4	9.0	2.54	0.601	0.601	930	0.048	0.025	0.073	0.54	0.08	87.9	85.3
	21	79.0	4.0	2.31	0.583	0.583	930	0.051	0.017	0.068	0.48	0.08	88.2	83.4
	22	81.1	5.0	2.59	0.696	0.663	930	0.063	0.015	0.078	0.56	0.08	88.3	85.7
	23	80.9	4.5	2.82	0.626	0.632	930	0.050	0.033	0.083	0.54	0.09	86.9	83.3
	24	79.4	7.0	2.57	0.645	0.676	930	0.048	0.027	0.075	0.62	0.12	89.0	80.7
	26	68.7	5.5	3.74	0.626	0.632	940	0.062	0.012	0.074	0.59	0.09	88.4	84.8
	27	80.7	5.5	2.90	0.654	0.666	930	0.047	0.024	0.071	0.60	0.08	90.2	86.6
	28	80.9	5.0	2.87	0.616	0.635	930	0.063	0.013	0.076	0.51	0.08	88.1	84.4
	29	79.2	7.5	2.95	0.585	0.610	920	0.054	0.023	0.077	0.44	0.07	87.5	84.2
	30	80.4	6.5	2.54	0.660	0.672	920	0.054	0.020	0.074	0.47	0.08	89.1	83.0
	31	81.1	5.0	2.03	0.710	0.697	920	0.058	0.022	0.080	0.58	0.08	88.6	86.3
Jan.	1	81.1	5.0	2.69	0.753	0.741	930	0.079	0.017	0.096	0.63	0.09	87.04	85.7
	2	81.3	3.5	2.84	0.716	0.691	920	0.064	0.015	0.079	0.46	0.06	88.56	87.0
	3	83.5	4.0	2.49	0.691	0.685	910	0.061	0.023	0.084	0.41	0.05	87.74	87.8
	4	81.0	5.0	2.76	0.740	0.672	920	0.065	0.023	0.088	0.49	0.08	86.90	83.7
	5	80.1	5.0	2.88	0.722	0.729	920	0.078	0.022	0.100	0.56	0.09	86.28	83.9
	6	81.8	4.0	2.75	0.735	0.729	920	0.054	0.016	0.070	0.42	0.06	90.40	85.7
	7	81.1	5.0	2.59	0.645	0.676	920	0.061	0.022	0.083	0.46	0.08	85.08	82.6
	8	82.3	3.5	2.47	0.657	0.608	920	0.053	0.016	0.069	0.48	0.06	88.65	87.5
	9	82.1	4.0	2.50	0.707	0.701	920	0.065	0.019	0.084	0.56	0.10	88.02	82.2
	10	79.2	4.5	2.72	0.707	0.688	920	0.061	0.019	0.080	0.51	0.07	88.68	86.3
	11	80.8	4.5	2.62	0.732	0.756	920	0.095	0.016	0.111	0.56	0.07	85.32	87.5
	12	80.7	4.0	2.59	0.756	0.750	920	0.081	0.027	0.108	0.58	0.09	85.60	84.5
	13	78.0	3.5	2.94	0.750	0.775	930	0.085	0.012	0.097	0.55	0.10	87.48	81.8
	14	83.0	3.0	2.66	0.856	0.818	950	0.109	0.022	0.131	0.70	0.12	83.99	82.9
	15	75.8	4.5	3.29	0.781	0.832	950	0.107	0.035	0.142	0.70	0.15	82.93	78.5
	16	76.5	3.0	2.69	0.732	0.725	940	0.074	0.024	0.098	0.62	0.13	86.48	79.0
	19	84.1	5.5	2.61	0.756	0.738	950	0.084	0.023	0.107	0.75	0.12	85.50	84.0
	20	73.2	4.0	2.92	0.787	0.795	960	0.083	0.023	0.106	0.67	0.12	86.32	82.1
	21	72.2	4.5	3.00	0.769	0.756	960	0.107	0.011	0.118	0.71	0.13	84.39	81.7
	22	72.3	4.5	2.76	0.756	0.775	980	0.089	0.024	0.113	0.52	0.10	85.42	80.8

TABLE II.—*Oxide Roast. Results from Laboratory Leaches. (Continued)*

Date		Rate of Feed Dry Weight	Per cent. Moisture in Feed	Per cent. Coal	Per cent. Copper in Feed	Per cent. Copper in Calcine	Average Temperature of Roast	Per cent. Copper as Cu_2S in Tails	Per cent. Copper as Ferrite in Tails	Per cent. Total Copper in Tails	Oz. Silver per Ton in Calcine	Oz. Silver per Ton in Tails	Extraction per cent. on Copper	Extraction per cent. on Silver
Jan.	23	71.7	4.5	2.53	0.763	0.781	1,000	0.063	0.026	0.099	0.65	0.17	87.32	73.8
	24	72.3	4.0	2.94	0.738	0.732	1,000	0.063	0.029	0.092	0.58	0.14	87.43	75.8
	25	71.5	5.0	2.83	0.692	0.717	1,000	0.048	0.033	0.081	0.70	0.18	88.70	74.3
	26	76.3	5.5	2.62	0.625	0.630	960	0.040	0.030	0.070	0.55	0.14	88.88	74.5
	27	83.8	5.0	2.70	0.655	0.643	950	0.043	0.021	0.064	0.48	0.08	90.05	83.3
	28	84.8	6.5	2.74	0.631	0.655	940	0.041	0.024	0.065	0.54	0.10	90.08	81.5
	29	83.4	5.5	3.28	0.576	0.643	950	0.046	0.018	0.064	0.58	0.12	90.05	79.3
	30	84.0	6.0	2.71	0.613	0.619	930	0.042	0.019	0.061	0.48	0.09	90.15	81.2
	31	80.6	6.0	2.82	0.668	0.625	930	0.053	0.012	0.065	0.57	0.10	89.60	82.5
Feb.	1	82.7	4.5	2.62	0.686	0.704	950	0.046	0.025	0.071	0.59	0.11	89.91	81.3
	2	80.0	6.0	2.33	0.695	0.775	970	0.057	0.025	0.082	0.66	0.12	89.42	81.8
	3	81.5	5.0	2.27	0.702	0.695	960	0.046	0.027	0.073	0.66	0.15	89.50	77.3
	4	78.5	7.0	2.50	0.691	0.683	970	0.046	0.027	0.073	0.78	0.19	89.31	75.7
	5	74.8	4.5	2.35	0.641	0.634	980	0.035	0.032	0.067	0.49	0.15	89.43	68.1
	9	81.5	4.5	2.56	0.653	0.671	980	0.038	0.021	0.059	0.48	0.08	91.21	83.3
	10	74.4	12.0	3.15	0.641	0.659	960	0.041	0.025	0.066	0.47	0.08	89.98	83.0
	11	81.0	5.0	2.44	0.666	0.691	920	0.046	0.020	0.066	0.67	0.12	90.45	82.1
	12	80.8	5.0	2.49	0.703	0.716	930	0.080	0.009	0.089	0.83	0.15	87.57	81.9
	13	81.7	4.0	2.59	0.635	0.660	930	0.053	0.019	0.072	0.68	0.11	89.09	83.8
	14	81.5	4.0	2.89	0.594	0.600	950	0.038	0.019	0.057	0.56	0.11	90.50	80.3
	15	81.2	4.5	2.92	0.643	0.643	960	0.041	0.023	0.064	0.67	0.14	90.05	79.1
	16	81.3	4.5	2.82	0.637	0.655	960	0.035	0.034	0.069	0.61	0.11	89.47	82.0
	17	79.9	5.5	2.78	0.625	0.637	930	0.043	0.030	0.073	0.54	0.09	88.54	79.6
	18	81.3	5.5	2.61	0.613	0.619	930	0.038	0.024	0.062	0.61	0.11	89.98	82.0
	19	80.0	6.0	2.73	0.588	0.606	920	0.040	0.023	0.063	0.53	0.09	89.60	79.2
	22	80.6	4.5	2.78	0.588	0.600	920	0.033	0.031	0.064	0.56	0.09	89.33	84.0
	23	80.6	4.5	2.87	0.662	0.643	920	0.044	0.024	0.073	0.58	0.09	88.65	84.4
	24	80.6	6.0	2.61	0.649	0.686	930	0.042	0.027	0.069	0.47	0.08	89.94	83.1
	25	89.6	5.5	2.77	0.637	0.662	930	0.042	0.023	0.065	0.38	0.04	90.18	89.5
Ave.		79.8	5.1	2.75	0.675	0.680		0.058	0.023	0.081	0.57	0.10	88.09	82.5

Recovery of copper, per cent	88.09
Recovery of silver, per cent	82.5
Pounds copper per ton in calcine	13.6
Pounds copper per ton in tailings	1.62
Pounds copper per ton recovered	11.98
Average feed per 24 hr. (tons dry wt.)	79.8
Average coal consumption, per cent	2.75

TABLE III.—*Results from Tank Leaches. Oxide-Chloride Roast*

Leach Number	Tons Calcine	Per cent. Copper in Feed	Per cent. Copper in Calcine	Top Half		Bottom Half		Total Tank		Washed Clean		Pounds Copper Recovered per Ton	Os. Silver per Ton in Feed	Os. Silver per Ton in Tails	Per cent. Extraction on Silver	Temperature Solution C°	Per cent. H_2SO_4 in No. 1 Solution	Per cent. NaCl in No. 1 Solution	Per cent. H_2SO_4 in No. 2 Solution	Per cent. NaCl in No. 2 Solution
				Per cent. Copper in Tails	Per cent. Extraction Copper	Per cent. Copper in Tails	Per cent. Extraction Copper	Per cent. Copper in Tails	Per cent. Extraction Copper	Per cent. Copper in Tails	Per cent. Extraction Copper									
1	309	0.580	0.540	.....	.....	.....	.....	0.202	65.2	.....	.....	7.56	0.55	0.23	5.82	20	3.03	4.70	5.10	4.90
2	391	0.569	0.500	.....	.....	.....	.....	0.151	73.5	.....	.....	8.36	0.55	0.17	69.1	20	3.72	5.16	6.15	7.17
3	390	0.560	0.506	.....	.....	.....	.....	0.140	75.0	.....	.....	8.40	0.55	0.15	72.7	20	4.70	7.05	6.33	8.50
4	380	0.592	0.526	.....	.....	.....	.....	0.113	80.9	.....	.....	9.58	0.55	0.20	63.6	65	5.20	8.26	7.40	9.09
5	385	0.541	0.491	.....	.....	.....	.....	0.097	82.1	.....	.....	8.88	0.55	0.08	85.5	65	5.30	5.00	8.90	7.80
6	360	0.536	0.491	.....	.....	.....	.....	0.111	79.3	.....	.....	8.50	0.55	0.12	78.2	65	5.95	6.93	8.60	7.96
7	457	0.575	0.540	.....	.....	.....	.....	0.135	76.5	.....	.....	8.80	0.55	0.19	65.5	65	4.30	7.50	7.50	8.40
8	370	0.680	0.654	.....	.....	.....	.....	0.136	80.0	.....	.....	10.88	0.55	0.17	69.1	65	4.30	7.90	9.10	7.00
9	368	0.683	0.631	0.077	88.7	0.085	87.6	0.081	88.1	.....	.....	12.04	0.55	0.08	85.5	65	2.90	7.00	6.00	6.90
10	224	0.701	0.662	0.092	86.8	0.106	84.9	0.099	85.9	.....	.....	12.04	0.55	0.10	81.8	65	3.40	8.20	5.00	7.50
Ave	....	0.602	0.554	0.085	87.8	0.096	86.8	0.127	78.9	.....	.....	9.50	0.55	0.15	72.9	....	.....	.....	.....	.....
							Oxide Roast													
11	370	0.616	0.629	0.097	84.8	0.072	88.5	0.085	86.5	0.081	87.1	10.88	0.53	0.15	71.7	65	2.13	8.91	4.68	7.53
12	410	0.686	0.681	0.104	84.7	0.124	81.8	0.114	83.3	0.101	85.2	11.34	0.53	0.26	50.9	65	2.90	8.16	7.13	7.59
13	430	0.636	0.648	0.091	85.9	0.157	75.8	0.124	80.9	0.105	83.8	10.48	0.56	0.15	73.2	65	2.18	11.06	6.50	9.62
14	400	0.713	0.710	0.129	81.8	0.146	79.4	0.138	80.6	0.122	82.8	11.44	0.54	0.42	22.2	65	2.58	10.30	5.33	7.17
15	375	0.718	0.713	0.082	88.5	0.087	87.8	0.085	88.1	0.077	89.2	12.56	0.51	0.08	84.3	65	5.02	11.61	6.60	9.18
16	390	0.761	0.759	0.096	87.4	0.105	86.2	0.100	86.8	0.095	87.5	13.18	0.66	0.18	72.7	65	4.18	7.11	9.20	9.84
17	410	0.656	0.660	0.071	89.2	0.100	84.9	0.086	87.1	0.075	88.6	11.48	0.56	0.17	69.6	65	2.82	8.44	18.00	8.72
18	385	0.677	0.686	0.097	85.9	0.139	79.7	0.118	82.8	0.101	85.3	11.36	0.60	0.21	65.0	65	2.85	7.72	6.50	8.80
19	395	0.654	0.670	0.086	87.2	0.122	81.8	0.104	84.5	0.097	85.5	11.32	0.60	0.15	75.0	65	3.78	7.29	7.05	8.59
20	395	0.616	0.629	0.086	86.3	0.115	81.7	0.101	84.0	0.096	84.7	10.54	0.58	0.15	74.1	65	4.65	10.15	7.05	9.50
Ave.	....	0.673	0.679	0.094	86.2	0.117	82.8	0.106	84.4	0.095	86.0	11.46	0.57	0.17	70.9	....	.....	.....	.....	.....

Precipitation of Copper from Solution at Anaconda

BY FREDERICK LAIST AND F. F. FRICK, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

Introduction

IN a leaching process, having obtained the copper in solution, the choice of the precipitation method is influenced by the following factors:

1. Availability of precipitant.
2. Adaptability to the leaching process.
3. Final product desired.
4. Regeneration of leaching solutions.
5. Fouling of solutions.

There are three general classes of precipitation methods:

1. The use of iron, scrap or sponge.
2. Electrolytic deposition.
3. The use of some gas or reagent by which the copper is obtained, usually in the form of an intermediate precipitate which requires further treatment. For example, the precipitation of the copper as Cu_2Cl_2 from chloride solutions, with SO_2 gas.

The use of iron is adapted to almost any process. It has the great advantage of extreme simplicity, and the recovery of the copper and any values present in a highly concentrated, easily treatable form. The disadvantages are the cost in isolated places, the uncertainty of the scrap-iron market, and the fouling of the leaching solutions. The latter is a serious factor, when a reagent such as salt is used in them, which makes necessary their re-use.

Where applicable, electrolytic deposition is very attractive. The copper is obtained in a form directly marketable, there is a regeneration of acid and no fouling of solutions. Unfortunately, chloride solutions do not lend themselves to electrolysis. An entirely satisfactory anode material has not yet been discovered, although magnetite, apart from being rather brittle, seems to answer the requirements fairly well.

The use of a method of the third class is involved in many special

leaching processes, and is highly attractive theoretically. Such methods usually involve regeneration of leaching solutions, no fouling of solutions, and the use of some cheap reagent, as lime, or a by-product of the process itself, or of some process such as smelting. Some method of this kind is undoubtedly that of the future where electrolytic deposition is not applicable.

The copper solution from the 80-ton leaching plant, in which the sand-treatment method now being installed on a large scale was developed, averaged Cu, 1.91; $Fe_2O_3+Al_2O_3$, 3.88; NaCl, 8.3 per cent.; Ag, 0.634 oz. The copper content of this solution cannot be increased much economically. The application of this solution to successive lots of roasted tailing, from which about 10 lb. of copper per ton can be recovered, would necessitate excessive washing of the tailing. The copper can be precipitated from this solution by means of scrap iron, the cement copper going directly to the blast furnaces or converters. The use of sponge iron is more attractive than the use of scrap iron, however. It can be made from calcines which contain about 50 per cent. iron. This makes the process independent of the scrap-iron market. But the use of iron necessitates the discard of about one-fourth of the solution each time to stop the accumulation of iron sulphate, and this leads to the loss of much valuable salt.

Owing to its high chlorine content the solution is not adapted to electrolysis. With a view to getting something better than scrap iron, experiments have been and are being conducted on precipitation by sulphuretted hydrogen made from matte (also from calcium sulphate), sponge iron from calcined concentrates, and sulphur dioxide from roasting-furnace gases.

In the following discussion we shall attempt to explain the operation, difficulties, advantages and disadvantages, and the results obtained with each of these methods.

Hydrogen Sulphide

Hydrogen sulphide was used as the precipitant in the work done on the experimental plant operated during the summer of 1912.

The precipitation tank, 10 by 10 by 7 ft., was equipped with an agitator and a system of lead pipes in the bottom for gas distribution. The generator was a lead-lined iron drum 2 ft. in diameter and 4 ft. high. It was built to stand 90 lb. pressure and fitted with air and steam connections. The iron sulphide was put in through a handhole in the top. The acid was introduced through an iron pipe by gravity. The gas passed through a lead main to the distributing pipes. When action ceased a plug was removed from the bottom of the tank, 90-lb. air applied and the residue blown out.

The copper sulphide was filtered in a small hand-made wooden filter press, the inside dimensions of the frames of which were 12 by 12 by $1\frac{5}{8}$ in. Ordinary filter paper backed up with heavy canvas was used. The copper sulphide was allowed to settle over night, the clear solution drawn off and the thickened residue drawn into a pressure cylinder, from which it was forced with 90-lb. air through a $\frac{1}{2}$-in. lead pipe to the press.

The absorption of the H_2S gas was very inefficient. The gas entered under considerable pressure and escaped in big bubbles through a depth of $3\frac{1}{2}$ ft. of solution. Some of the iron sulphide, which was the commercial kind, was of very poor quality. It was found that when ground to 4 mesh the rate of evolution of the gas was about right.

A charge was 400 lb. of water, 100 lb. of 4-mesh iron sulphide, and 130 lb. of 66° Baumé acid which was slowly added. The acid strength on starting was about 24.5 per cent. and averaged 7.5 per cent. at the end; 2.4 lb. acid per pound of copper was required for the generation of the H_2S. The efficiency of the acid used in the generation was about 62 per cent. The efficiency of the FeS in the laboratory was 80.2 per cent., but in the plant only 54 per cent. was realized, due to poor absorption of the gas.

About 100 lb. of matte was made by fusing in proper proportions a heavy pyrite ore, an oxidized iron ore, and lime rock, in graphite crucibles. This matte decomposed readily and was 97.2 per cent. as efficient as the commercial FeS.

Analysis of Matte and FeS

	Matte, Per Cent.	FeS, Per Cent.
FeO	78.8	81.2
S	31.1	28.6
Cu	2.23	....
Ins	0.8	0.4

The filtration gave some trouble, due to defects in the press. When the press was working well the cakes were of about the consistency of cheese and assayed: Cu, 58; S, 29.0 per cent.; Ag, 69.5; Au, 0.04 oz. The cakes carried 48 per cent. moisture.

With a generator having an agitating device, and a properly designed absorber, there is no doubt but that a high efficiency could be obtained by the use of this process. The reaction between $CuSO_4$ and the H_2S liberates 1.55 lb. acid per pound of copper precipitated, and the reduction of the ferric salts also liberates some acid.

The advantages of this process are,

1. Regeneration of acid.
2. No fouling of solutions.
3. A product which is high grade and can be readily worked into the smelter process.

4. The matte used in generating the H_2S is enriched by removal of its iron and sulphur.

Another method of generating H_2S which was considered, but not experimented with in connection with the present problem, consists of an adaptation of the Chance process, which is most successfully used for working up the so-called "soda waste," resulting from the manufacture of soda by the Le Blanc process. "Soda waste" is largely composed of calcium sulphide (CaS). This is finely ground and made into a thin mud with water and is then decomposed with carbon dioxide, the reaction being as follows: $CaS + CO_2 + H_2O = CaCO_3 + H_2S$.

Experiments were made on this line by one of the writers several years ago and gave quite satisfactory results. The calcium sulphide (CaS) was made by reducing gypsum ($CaSO_4$) with coal in a shaft furnace similar to a lime kiln. The gypsum and coal were formed into bricks with water. It was found that a very fair quality of calcium sulphide could be made in this manner, a yield of about 75 per cent. of the theoretical being obtained.

The calcium sulphide was decomposed by means of gases taken from the reduction furnace. These averaged about 14 to 16 per cent. CO_2 and answered the purpose very well. The calcium sulphide absorbs the CO_2 with avidity. No heat is required. A train of five cylinders, 1 ft. in diameter by 3 ft. tall, constructed of galvanized iron and connected by pipes and valves in such a way that any cylinder could be made the first (or last) of the series, was used for the decomposition of the CaS. The CO_2 gas was forced through the cylinders by means of a small compressor. The gas on leaving the last CaS cylinder was conducted to a similar series of wooden cylinders containing copper solution, no intermediate pumping of the gas being required. The absorption of the H_2S was perfect. No odor whatever was perceptible at the outlet of the last cylinder. The conversion of the calcium sulphide into calcium carbonate by the carbon dioxide gas was also perfect. No odor of H_2S could be detected when the residue from the tanks was dissolved in dilute hydrochloric acid.

Sponge Iron

In June, 1913, a sample of a concentrate high in iron sulphide was roasted to a calcine assaying: SiO_2, 4.3; FeO, 72.2; S, 4.2; Cu, 6.2; Fe, 56.2 per cent.

Two parts of this calcine were mixed with one part of fine Diamondville coal, sealed in a graphite crucible and heated to about 1,700° F. for a couple of hours. The residue had a copper equivalent of 1.77. (The "copper equivalent" is the grams of sponge iron required to precipitate 1 g. of copper from a 1 per cent. copper solution prepared from crystallized copper sulphate.) A series of crucible tests followed, which gave suffi-

ciently good results to warrant the collection and roasting of about 125 tons of the special high iron sulphide for further experiment.

The roasting was done in a 16-ft. six-hearth MacDougall furnace at an average rate of 16.3 tons per day. Oil burners were used on the lower hearths to maintain the temperature so that the roast could be carried below 4 per cent. in sulphur.

Assays of Concentrates and Calcine

	Concentrates Per Cent.	Calcine Per Cent.
Cu	5.7	....
FeO	49.0	....
Ins	7.0	14.0
S	....	3.4

A retort furnace was built near the leaching plant and operated during October and November, 1913. The equipment consisted of a three-retort coal-fired furnace, an elevation of which is shown in Fig. 1, and three air-tight cars for cooling the sponge. The retorts, similar to those used in gas making, were 8⅝ ft. long, 15 in. deep, and 2 ft. wide inside, had 3 in. thick walls, and cast-iron fronts with gas connections. The fine reduced iron reoxidized so easily that air-tight cars were necessary for cooling.

The charge consisted of a mixture of 30 parts ground Diamondville coal and 70 parts calcine. The first lot was prepared by grinding the coal, mixing with the calcine and grinding the mixture so that 96 per cent. passed 40 mesh. Later it was found unnecessary to grind the calcine. The coal was ground in a Hardinge mill so that 90 per cent. passed 60 mesh.

The first charge, only one muffle used, was left in for 24 hr. and gave a copper equivalent of 2.3. This was continued for a couple of weeks. The temperature was raised as fast as possible and the pyrometer reading was at 1,600° for at least 4 hr. before discharging. The pyrometer rod was not stuck down in the mixture, but was above it. The results from these charges were uniformly good, the copper equivalent varying from 2.2 to 3.0 g. iron per gram of copper. No special precautions were taken at any time in charging or discharging. The samples of material were taken with a pipe sampler from a car of cooled material. The charge was always dampened to prevent too much dusting. The charges were about 450 to 500 lb. of original mixture. These gave on an average, 275 to 300 lb. of "iron." One charge during this time was rabbled every half hour for 9 hr. and the temperature worked up as fast as possible. It was discharged at a temperature of 1,500° F. and gave a copper equivalent of 2.7 g. iron per gram of copper.

At the end of the first two weeks it was deemed advisable to try to

get more material through the furnace, and a scheme was worked out to give each charge 8 hr. and work them off faster. This proved unsatisfactory. It was not possible to obtain more than 1,300° temperature and the resulting iron was very poor. About this time the muffles began to crack and let air in. This naturally resulted in the production of a very poor grade of iron.

The best iron which was made under the foregoing conditions was a black, dusty material. It contained considerable coke and was quite light. The coarser calcine gave as good results as the ground material. None of the pyrometer readings was over 1,620°. In all cases the pyrome-

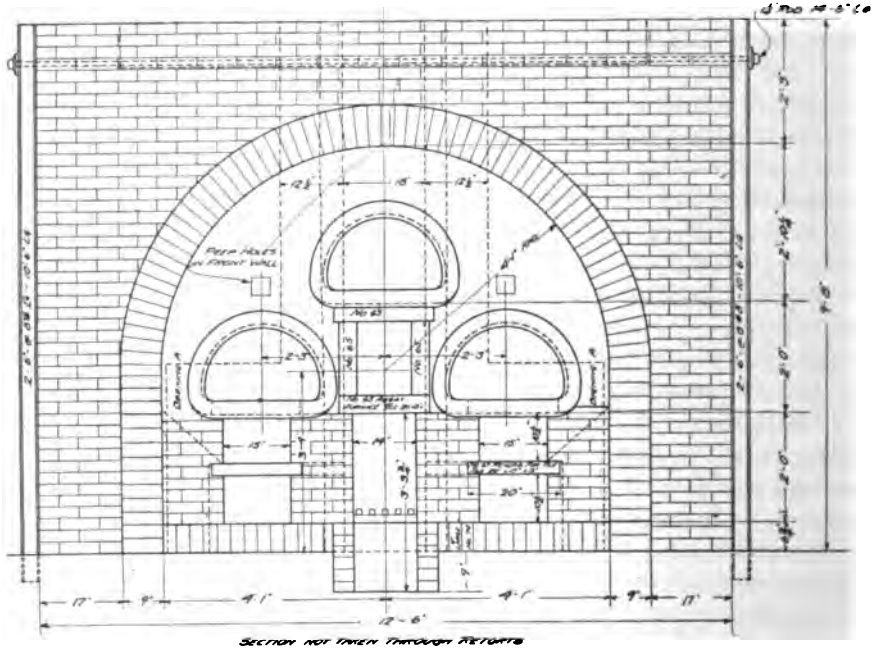

FIG. 1.—ELEVATION OF SPONGE IRON RETORT FURNACE.

ter tube was over and not buried in the mixture. Toward the end of the experiments, the muffles had quite a cake in the lower corners and on the bottom, which seemed to be more or less fused to them, and was impossible to clean out.

It was early recognized that while a good sponge iron could be made in retorts, the fuel efficiency was low and the life of the retorts limited. So it was decided to try reduction in a specially rebuilt MacDougall furnace. The furnace was so equipped that hot partly roasted calcine from another furnace was fed to the first hearth by a steel conveyor. The roasting was completed and the calcine preheated to 1,700° by oil

burners on the third hearth. The three lower hearths were equipped with oil-fired muffle floors. Fine coal was fed in on the fourth hearth. The plan was to obtain a mixture of the preheated calcine and fine coal on the lower hearths under strongly reducing conditions such that the reduction would be accomplished. The oil burners directed into the lower hearths worked all right, but those into the muffles, due to the necessarily restricted size of the muffle, localized the flame too much, melting the bricks in the direct path of the flame and depositing carbon on the other portions.

The oil burners were removed and coal firing was tried. Two fire boxes were installed on the third floor, the fine coal being fed on the fourth floor. The addition of this cold coal lowered the temperature to such an extent that the calcine did not reduce, and on increasing the temperature the coal burned off. It seemed impossible to get a high temperature and a reducing atmosphere at the same time on the lower hearths. It was then decided to add another fire box. At first it was connected to the fifth hearth muffle only, but finally to both the fourth and fifth hearth muffles. While increasing the heat in these hearths, the muffles were poor conductors of heat and the piers and floors above burned out. Gas analyses made on the fifth and sixth hearths showed CO_2, 15; O_2, 2.7; CO, 0.7 per cent., the temperature in the fifth hearth being 1,240° F. The best gas sample showed only 2.0 per cent. CO on the fifth hearth and this for only a short time.

Fine coke was next tried and had the same effect of cooling the calcine. Though added on the third hearth, it practically all dropped down to the fourth hearth before mixing with the calcine. By laboratory experiments it was shown that coke did not have the same reducing action on calcines as coal or charcoal under the same conditions and its use was discontinued. No appreciable reduction was obtained during this period. The product made by the furnace was magnetic, but showed no signs of metallic iron and would not precipitate any copper.

While using two fire boxes the coal burned was 4,000 lb. per 24 hr. When the three were in operation it required 7,500 lb. per 24 hr. The special iron concentrates fed to MacDougall No. 62 at a rate of 26 tons per 24 hr. averaged: Insol., 7.1; FeO, 47.6; and S, 46 per cent. They were roasted to 5 per cent. S.

Owing to the cooling effect of adding the coal dust to the calcine in the furnace, feeding a mixture of coal dust and calcine thoroughly mixed was tried. A mixture of two-thirds calcine and one-third coal dust by weight was fed into the top of the furnace. No difficulty was experienced in obtaining a high heat on the three upper hearths, but the lower hearths were cold. No reduction was obtained, due, probably, to unavoidable leakage of air.

In January, 1914, it was decided to start a new series of experiments. A series of laboratory experiments had been made in December using

Diamondville coal, Sunnyside coke, and charcoal as reducing agents. Equally good results were obtained from the coal and charcoal, but practically no reduction from the coke at the temperatures used. The significance of these results was not appreciated at the time, as will be explained later.

The retort furnace was remodeled, a scrap cast-iron hydraulic cylinder 8½ ft. long and 18 in. in diameter, with a hinged door, being installed. This retort was operated for a couple of weeks with a mixture of 30 parts fine Diamondville coal and 70 parts calcine, with indifferent results.

In the meantime it happened that a crucible run was made, in the laboratory, using lumps of coal about thumb size instead of fine coal. To our surprise, excellent results were obtained. A new line of experiments was started in which calcine was heated and then mixed with coarse coal. A charge of calcine was heated to 1,500° in the retort and coarse coal was charged and rabbled in. The results were poor because of the difficulty of rabbling the coarse coal down into the fine calcine. However, some reduction was obtained.

The great disadvantage of the retort method is the low fuel efficiency. This was especially true when Diamondville coal, 38 per cent. volatile combustible matter, was used. The gas formed carried off the heat as rapidly as applied and the temperature could not be raised above 1,000° F. until it was driven off. It was known that efficient reduction did not take place under 1,500° F., after all the gas was driven off; also that while blast-furnace coke was not a reducer at this temperature, charcoal was. So it was suggested that the coke resulting from the coal must be the reducer and that the hardness and character of the coke must be factors in its efficiency.

Diamondville coal assayed in 1912: volatile combustible matter, 38.38; fixed carbon, 45.28; ash, 10.01 per cent. It is not a coking coal, in the commercial sense, coke resulting from it being very soft and friable.

A charge of fist-size Diamondville coal was coked in the retort at 1,400° F. A charge of coke was ground, mixed with calcine, and reduced in the retort with indifferent success. The next day the remaining coke, uncrushed, was charged with calcine into the retort. Conditions seemed to be just right this day and an excellent product resulted. The residue was a gray spongy mass mixed with coarse unconsumed coke and had a copper equivalent of 1.4. It was now decided to preheat coke and calcine under efficient conditions and mix them in an air-tight container.

The retort was shut down and a series of 40 crucible tests made in the laboratory to determine conditions of time, temperature, per cent. of reducer, hardness of coke, possibility of mixing hot, and other details. The results of these experiments showed that if 15 parts of Diamondville coke and 85 parts of calcine could be preheated to 1,800° F., mixed, and maintained at this temperature for an hour, the resulting residue would have a copper equivalent of 1.5.

A double crucible furnace to hold two No. 50 graphite crucibles was built. Oil burners were used for heating. The mixer was a sheet-iron shell 2 ft. deep and 20 in. in diameter lined with 4½ in. of fire brick. The mixer was provided with an air-tight cover and three tuyères, and placed on trunnions so it could be dumped. Coke was burned in the mixer to preheat it, air being supplied through the tuyères. When the charge was ready the coke was dumped from the mixer and the tuyères closed.

Charges of calcine and coke were heated in the crucibles to between 1,700° and 1,800° F. and dumped together into the mixer. There was too much cooling in this manipulation and poor results were obtained. Putting the coke in the mixer and heating it by burning a part of it, using air through the tuyères, was tried. This failed, due to the difficulty of getting the coke heated to a uniform temperature throughout. However, a copper equivalent of 3.2 was obtained from a picked sample.

It was then decided to build two small hand-rotated, oil-fired, brick-lined cylinders for preheating, from which the charges could be very quickly drawn into the mixer. The soft Diamondville coke was heated in one cylinder and the calcine in the other cylinder to 1,800° F. The charges were so small, however, that when they were transferred to the mixer they lost about 300° F. in temperature. The resultant mixture, 1,500° F. at the start, cooled rapidly and very poor results were obtained.

Heating a mixture of the coke and calcine in one cylinder to 1,800° and then quickly sealing the ends, gave a product with a copper equivalent of 2.35. This was so promising that a larger furnace of the Brückner type was made by lining a section of a White Howell furnace, 5 ft. in diameter and 12 ft. long, with 4½ in. of fire brick about the middle and a 30-in. brick wall in each end, with a 12-in. opening in one end for an oil burner and an 18-in. opening in the other for a stack connection. A 6-in. charging and discharging door was put in one side midway between the ends, and a power drive arranged which gave 1.28 rev. per minute. This gave a furnace 4 ft. 3 in. in diameter inside and 7 ft. long, with a capacity of 2,000 lb. of charge. Fig. 2 is a photograph of this furnace, including the oil burner.

The sheet-iron box around the middle of the furnace is stationary and makes a reasonably air-tight space in which the sponge iron can be discharged without excessive oxidation. The discharging door revolves inside the box and the iron falls into a rapid stream of water in a launder below, where it is quenched and washed into a collecting box. Complete data on this furnace have not been obtained. However, the results obtained so far are satisfactory and no difficulties are apparent which will prevent the development of the process to a commercial basis.

The furnace is operated as follows:

1,400 lb. calcine are charged and heated with a fuel-oil flame to about

1,300° F. This requires about 1¼ hr. In a commercial plant the calcine would be drawn hot directly from the MacDougall hoppers to the furnace. About 600 lb. of coal are then shoveled in through the front, in small lots. The furnace continues to revolve and in about ¾ hr. after starting to charge the coal the hydrocarbons are burned off. The oil flame is again started. In 1¾ to 2 hr. the charge is up to 1,680° to 1,700° and reduction is complete. The discharging door is removed and the charge quenched.

We are making sponge iron with a copper equivalent of 1.5 on samples screened through a 14-mesh screen. The coal used amounts to 40 per cent. of the weight of the calcine. It is probable that this percentage

FIG. 2.—SPONGE-IRON FURNACE.

can be reduced. Coke to the extent of 25 per cent. of the weight of the coal is recovered by screening the product discharged from the furnace. This coke can undoubtedly be used again. The quenching works well, the quenched product averaging 1.7 copper equivalent. The best results are obtained at a temperature of 1,680° to 1,700° F. If the temperature is carried much above 1,700° F. there is a tendency for the charge to nodulize. The fuel consumption for heating, 20 gal. fuel oil to a ton charge, is very reasonable when it is considered that the process is intermittent, the scale is small, and the apparatus is still in the experimental stage. We expect to present a paper later giving a full account of this process.

Sulphur Dioxide Precipitation

When cupric copper in solution is reduced with SO_2, in the presence of chlorides, there is a precipitation of cuprous chloride, the amount of

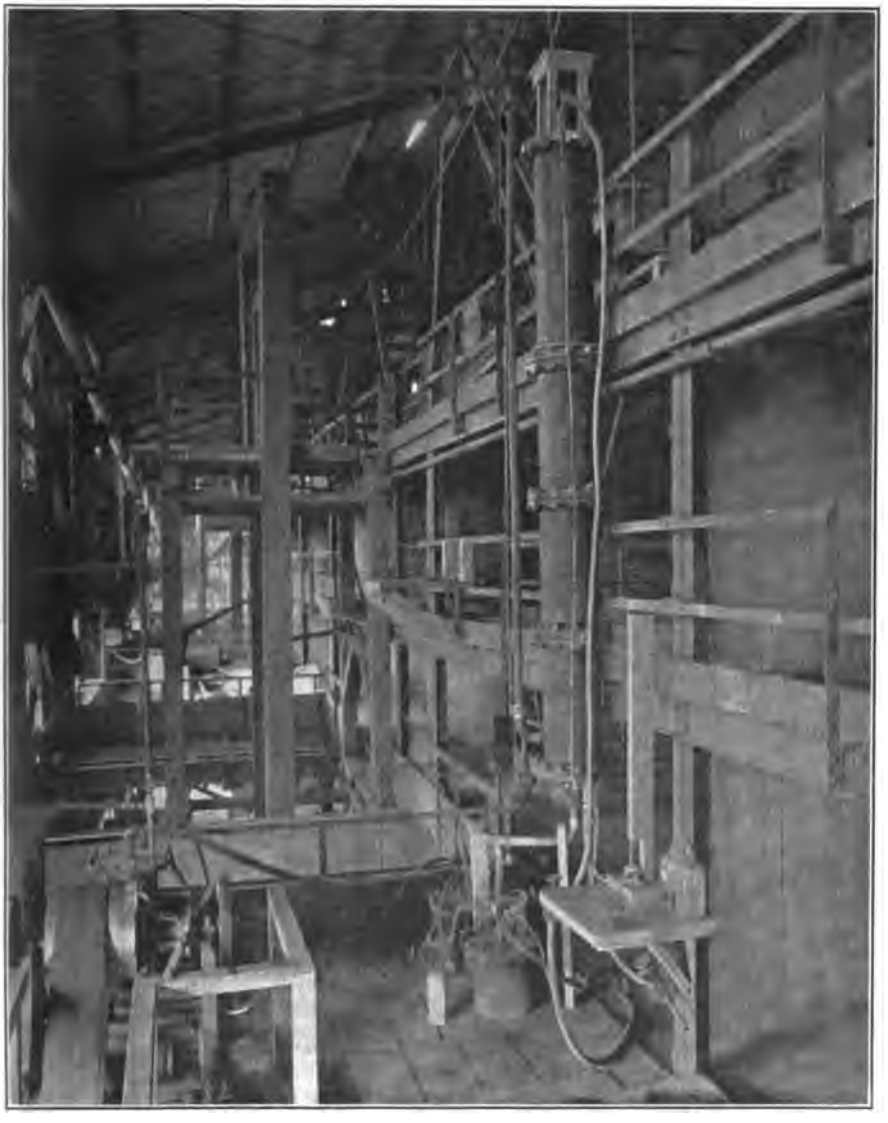

FIG. 3.—TWO-TON SULPHUR DIOXIDE PRECIPITATION PLANT.

precipitation depending on the solubility of the cuprous chloride in the solution. The reaction is, $2CuCl_2 + SO_2 + 2H_2O = Cu_2Cl_2 + H_2SO_4 + 2HCl$.

According to this reaction the equivalent of 1.54 lb. of H_2SO_4 is regenerated per pound of copper reduced. Actually, from our solutions, 2 to 2.5 lb. of acid per pound of copper reduced are regenerated, due to reduction of ferric salts and a catalytic action of the cuprous chloride. Either pure SO_2, gaseous or liquid, or a gas containing 10 per cent. SO_2 by volume, can be used for the reduction. A gas containing less than 8 per cent. SO_2 by volume does not give satisfactory results. The most satisfactory conditions for the reduction and precipitation of the copper, so far as we have determined them, are saturation of the cold solutions by passing the SO_2 gas through at 15 lb. pressure per square inch, heating to boiling under 20 lb. pressure per square inch and then cooling to 60° to 70° F. The

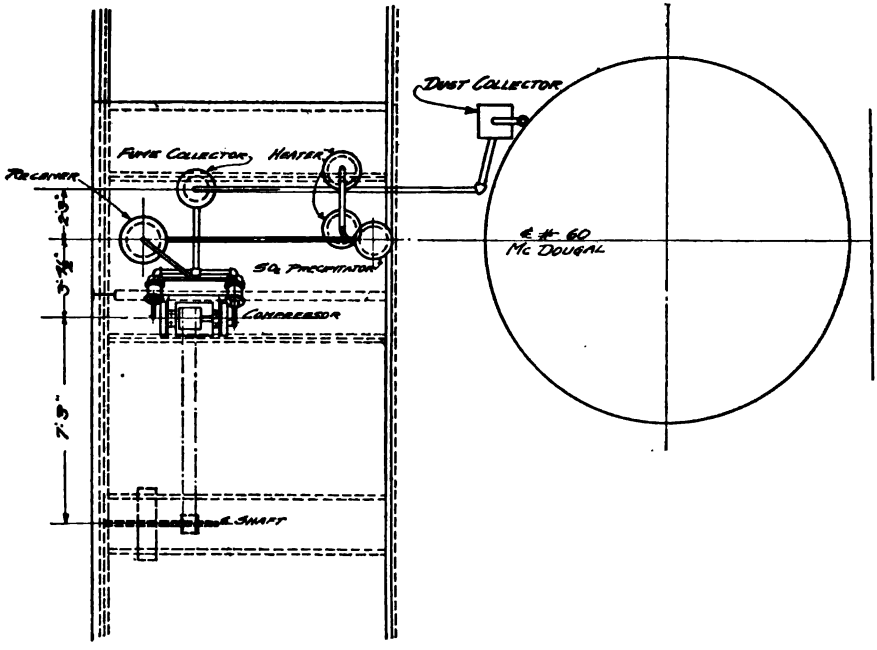

FIG. 4.—PLAN OF 2-TON EXPERIMENTAL SULPHUR DIOXIDE PRECIPITATION PLANT.

cuprous chloride separates as a heavy white crystalline precipitate which settles readily.

Three and one-half tons of the 2 per cent. Cu solution carrying 8.5 per cent. NaCl have been precipitated in 500-lb. lots. An average of 80 per cent. of the copper and 100 per cent. of the silver was precipitated, giving a tail solution carrying 0.4 per cent. Cu and 4.0 per cent. H_2SO_4, the cuprous chloride being soluble in this 8.5 per cent. NaCl solution to the extent of 0.4 per cent. Cu.

The first experiments were made in a lead-lined iron auto-clave, holding about 700 cc.. The solution was saturated under 10 lb. pressure

per square inch with pure SO_2 gas, obtained from cans of liquid SO_2, transferred to the auto-clave, and heated to boiling under 10 lb. pressure per square inch. On a leaching-plant solution carrying 1.5 per cent. Cu a 71 per cent. precipitation was made.

The copper solution used in the following experimental work contained: Cu, 2.0; $Fe_2O_3 + Al_2O_3$, 4.6; NaCl, 4.8 to 8.5; H_2SO_4, 0.5 per cent.; Ag, 0.45 oz.

A lead-lined pressure tank 1 ft. in diameter and 2 ft. deep, with a lead heating coil, was used. The copper solution from the 2,000-ton sands-leaching plant will have about this analysis and carry 8.5 per cent. NaCl. The solution was saturated under pressure, which was maintained during the heating stage. At first pure SO_2 was used. Later 10 per cent. SO_2 gas made from pure SO_2 and air was tried. After the details of manipulation were worked out a 90 per cent. precipitation was made with no trouble. Following are the data of one of these runs:

Volume solution, liters	26.5
Per cent. Cu in head solution	2.0
Per cent. NaCl in head solution	4.8
Per cent. Cu in tail solution	0.2
Per cent. acid in head solution	0.5
Per cent. acid in tail solution	4.0
Per cent. SO_2 in gas	10.0
Saturation time in hours	3.0
Heating time in hours	4.0
Final temperature in degrees F	195.0
Saturation pressure in pounds	15.0
Heating pressure in pounds	20.0
Per cent. precipitation	90.0

NOTE.—Later by a change of steam connections the heating was accomplished in ½ hr.

Experiments were made on the operation of a MacDougall roaster to give a 10 per cent. gas. The furnace was run hot on a heavy feed of undried fine concentrates. The gas, obtained without difficulty at 10 per cent., was drawn from the second hearth by a small compressor, and compressed to 90 lb. in a 5-cu. ft. oxygen cylinder. This gas gave as good results as the mixture of pure gas and air.

The success of the experiments mentioned warranted a trial of the method on a larger scale, 500 lb. to the charge. The installation shown in Fig. 3 was put in at the MacDougall building. It consists of a water-jacketed cooling pipe from the furnace, a dust catcher, a bag box, a small compressor, a receiver, the absorber, and two heaters. The general plan is shown in Fig. 4. A section of the absorber and one of the heaters is shown in Fig. 5. The absorber is made of sections of lead-lined 1-ft. diameter pipe with perforated diaphragms between sections. It holds about 500 lb. of solution. The heaters are lead-lined cylinders 4 ft.

long and 1 ft. in diameter. A woolen bag is used to remove the fine dust and fume. The residue in the bag consists of fine dust, elemental sulphur

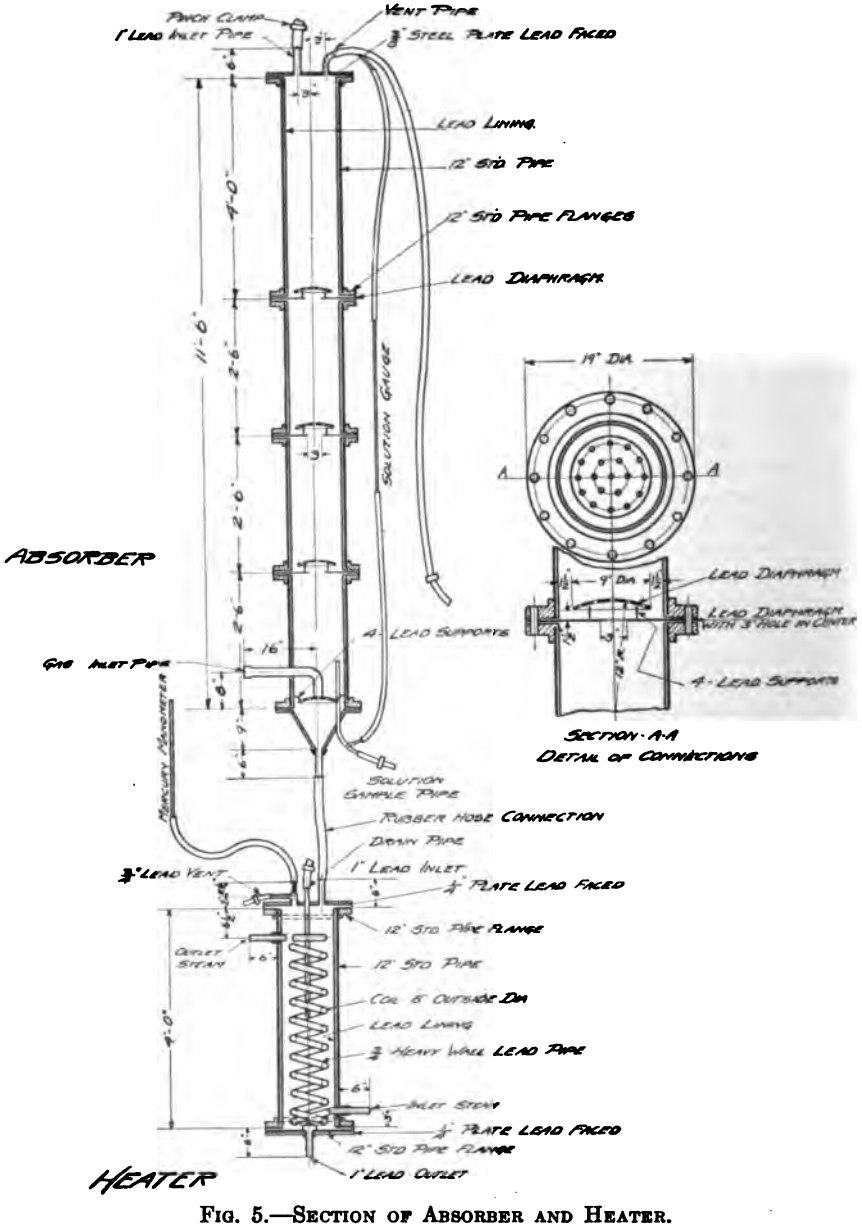

FIG. 5.—SECTION OF ABSORBER AND HEATER.

arsenic, and sulphuric acid, and is very corrosive, the bags lasting only about 6 hr.

The absorber is charged with about 500 lb. of 2 per cent. copper solution. The SO_2 gas is forced in under the lowest diaphragm. A pressure of 15 lb. per square inch is maintained by regulation of the vent. The gas has been passed through at rates varying from 0.8 to 6.1 cu. ft. per minute. The absorption of the SO_2 gas is almost complete for the first hour, and then gradually decreases to about 60 per cent. The solution changes from the blue cupric color to a greenish-brown color, and some Cu_2Cl_2 is precipitated. Samples are taken every hour, and after 2.5 to 3 hr. a small portion boiled in an Erlenmeyer flask and cooled has a characteristic appearance when the reduction is completed, which we have learned from experience indicates that the charge is ready for heating. The charge is drawn off in 125-lb. lots to the heaters. The heating time averages 35 to 40 min. The pressure is maintained at 20 lb. After 35 to 40 min. the pressure is gradually reduced and the charge allowed to boil several minutes. The four heats from one absorber charge are drawn into a barrel, and after cooling over night the supernatant solution is sampled for copper.

The first 18 runs were made on a solution running 2 per cent. Cu and 4.8 per cent. NaCl. An average precipitation of 90 per cent. of the copper was obtained. Then it was decided to increase the salt content of the solution to 8.5 per cent., the percentage which will be carried in the solutions from the 2,000-ton sands-leaching plant.

Table I shows the data of 14 runs. In all these runs, 500 lb. of copper solution assaying 2 per cent. Cu, 8.5 per cent. NaCl, and 0.5 per cent. H_2SO_4 was used. The saturation pressure was 15 lb. and the heating pressure 20 lb. per square inch. The absorption of the SO_2 gas varied from 60 to 75 per cent. The average copper content of the tail solutions was 0.36 and the average precipitation 81.7 per cent.

A series of runs were made with varying salt percentages. Other conditions were about the same as those given in the table. Extra salt was added to the regular leaching plant solution to bring it up to the desired salt content.

NaCl, Per Cent.	Cu in Tail Solution, Per Cent.	Precipitation, Per Cent.
9	0.42	79.0
9	0.42	79.0
8	0.32	84.0
7	0.26	87.0
6	0.18	91.0

The best precipitation ever obtained was on a laboratory experiment. The tail solution carried 0.14 per cent. Cu and the precipitation was 93.0 per cent.

The operation of this plant was highly satisfactory. The only trouble experienced with the plant proper was the failure of the heater linings.

TABLE I.—*Precipitation of Copper by Sulphur Dioxide.*

Run No.	19	20	21	22	23	24	25	26	27	28	29	30	31	32	Avg.
Avg. per cent. SO_2 in head gas	8.1	.....	9.0	7.2	9.1	8.2	7.0	9.7	.....	8.8	7.8	8.8	11.0	8.0	8.558
Cu. ft. gas per minute	.....	.....	6.0	6.0	6.0	6.0	6.0	.....	.....	6.0	6.0	4.5	6.0	6.0	5.85
Gas passed, hours	2.5	.....	2.5	2.8	2.5	2.25	2.5	2.5	.....	2.5	2.25	3.0	2.25	2.5	2.504
Vol. gas passed, cu. ft.	.....	.....	900.0	1,008.0	900.0	810.0	900.0	.....	.....	900.0	810.0	810.0	810.0	900.0	874.8
Vol. SO_2 passed	.....	.....	81.0	78.0	82.0	66.0	63.0	.....	.....	79.0	63.0	71.0	89.0	72.0	74.4
Avg. heating time, minutes	45.0	45.0	45.0	45.0	45.0	45.0	45.0	35.0	35.0	30.0	30.0	30.0	30.0	30.0	38.21
Per cent. Cu in tail solution	0.35	0.40	0.40	0.28	0.35	0.38	0.35	0.40	0.40	0.30	0.37	0.40	0.32	0.40	0.364
Per cent. H_2SO_4 in tail solution.	4.25	3.1	3.9	4.2	4.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	3.91
Per cent. precipitation of Cu	82.0	80.0	80.0	86.0	82.5	81.0	83.0	80.0	80.0	85.0	81.0	80.0	84.0	80.0	81.75

The heaters were originally lined with old 6-lb. lead, and both failed. The first one which failed, after relining with new 8-lb. lead, served for 48 heats and then showed no appreciable deterioration.

Reduction of Cuprous Chloride

There are three methods of reduction of the cuprous chloride which have been worked out to some extent and which seem capable of development to a commercial basis.

1. Precipitation with sponge iron.
2. Reduction with coke with limestone present to flux and hold the chlorine.
3. Electrolytic reduction.

Precipitation with Sponge Iron

Sponge iron, when mixed with cuprous chloride 40 to 50 per cent. solids, gives a rapid and complete precipitation, with the evolution of considerable heat. The reaction is, $Cu_2Cl_2 + Fe = 2Cu + FeCl_2$. It is evident from this reaction that only one-half as much iron is required to precipitate the copper from cuprous chloride as from a cupric salt. A sponge iron with a copper equivalent of 2.30 on a copper sulphate solution gave a copper equivalent of 1.20 on cuprous chloride. The cement copper from the precipitation of copper from cuprous chloride with sponge iron is rather granular, settles readily and washes easily. A sample made from sponge iron with a 1.20 copper equivalent on cuprous copper assayed 56 per cent. Cu and 0.1 per cent. Cl.

The precipitation of the copper from solutions by the SO_2 method followed by the treatment of the cuprous chloride with sponge iron makes an effective combination. Concentrates may be roasted in a suitable furnace to make the required SO_2 gas, and the hot calcine drawn directly from the hoppers beneath the roasting furnace into the revolving furnace for reduction to sponge iron. In a suitable furnace the concentrates can be roasted to under 4 per cent. S and 10 per cent. gas produced. It would figure about as follows: 100 tons concentrates assaying, Cu, 5.3; insoluble, 6.4; FeO, 48.6; Fe, 37.8; S, 44.8 per cent., will roast to a calcine assaying, Cu, 8.4; insoluble, 10.6; FeO, 62.6; S, 3.7 per cent. About 40 tons sulphur or 80 tons SO_2 are available. About 60 per cent. of this SO_2, or say 50 tons, can be absorbed in precipitation of the copper. The reaction is, $2CuCl_2 + SO_2 + 2H_2O = Cu_2Cl_2 + H_2SO_2 + 2HCl$. It is evident from this reaction that 50 tons SO_2 will precipitate about 100 tons copper.

The original concentrates contain 37.8 tons iron, of which 80 per cent. is available, for precipitation of copper, in the form of a sponge iron with

a copper equivalent of 1.5 to 1.6 on copper sulphate solution and 0.8 to 0.9 on cuprous copper. Cuprous copper requires 0.44 lb. pure iron per pound of copper for precipitation. $37.8 \times 0.80 \div 0.44 = 68.7$ tons copper precipitated. This gives a comfortable margin of SO_2 for precipitation and any excess can be easily wasted. The above calculation is rather rough, because, as before mentioned, the complete data of the sponge iron process are not yet worked out, but it is probable that it is not far off.

The combination of the methods has these advantages:

1. The conversion of a large part of the SO_2 from the roasting process directly into sulphuric acid in the leaching solutions, accompanied by the precipitation of the copper in a concentrated form, from which it can be recovered by precipitation with the sponge iron resulting from the reduction of the calcine.

2. The use of only one-half as much iron to precipitate the copper from the cuprous form as from the cupric, and hence the return of only one-half as much iron into the leaching solutions to foul them.

3. The precipitation of the copper in a concentrated condition by a simple process, in a form easily washed and handled.

4. The possibility of the treatment of the ferrous chloride which results in such a way that the valuable chlorides can be returned to the leaching solutions unaccompanied by iron.

Reduction with Coke in the Presence of Limestone

By this method the cuprous chloride is directly reduced to metallic copper, by a furnace process, with the production of a calcium chloride slag from which the chlorides may be leached and returned to the leaching process. Cuprous chloride is very volatile at high temperatures. However, by having fine limestone about 10 to 15 per cent. in excess of the theoretical requirements, in intimate mixture with the coke and cuprous chloride, it is possible to accomplish reduction with a loss of not more than 5 or 6 per cent. of the copper.

A mixture of 100 parts of slightly moist cuprous chloride, 65 parts limestone, and 10 parts coke was melted in a No. 50 graphite crucible and the melt poured into a sand mold. The button assayed 97 per cent. and the slag 0.88 per cent. Cu; 93.1 per cent. of the copper was recovered in the button and 1.4 per cent. in the slag, making a total recovery of 94.5 per cent. The slag, which carried 86 per cent. of the original chlorine in the cuprous chloride, and assayed 35 per cent. Cl, was leached for 2 hr. with warm water; 56.6 per cent. was soluble and the residue assayed 1.6 per cent. Cl. This indicated a recovery of 98.7 per cent. of the chlorine in the slag and 84.9 per cent. of the original chlorine in the cuprous chloride. This recovery was probably low, due to loss of slag in cleaning up. The insoluble, consisting mostly of lime and carrying some copper, settled

readily and could be worked up in the smelter process for its copper value without trouble.

Briquettes were made of a mixture of 100 parts moist cuprous chloride, 75 parts limestone, 10 parts coke, and 5 parts cement, by hand pressure, and with a testing machine at 2,500 to 5,000 lb. per square inch, in a die 3 in. in diameter. Excellent briquettes resulted in all cases, assaying about 26 per cent. Cu. A charge of briquettes was melted down in a small reverberatory about 12 in. wide and 20 in. long, using the flame of an oil burner for heating; 94.1 per cent. of the copper was recovered, 87.6

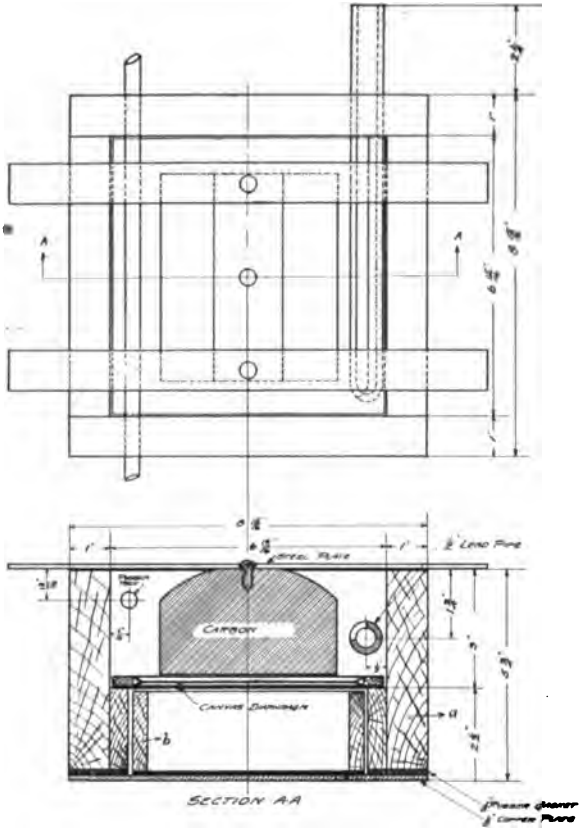

FIG. 6.—PLAN AND CROSS-SECTION OF 6-IN. REDUCTION CELL.

per cent. in a button and 6.5 per cent. in the slag. A charge of briquettes was also fused in a small cupola; 93.9 per cent. of the copper was accounted for, 90.8 per cent. in the button, which assayed 96.1 per cent. Cu, and 3.1 per cent. in the slag, which assayed 1.65 per cent. Cu.

In all cases there was considerable fume given off and the flame had a characteristic bluish color. It is very probable that with a suitable

installation 95 per cent. of the copper could be recovered in metallic form and in the slag, and that 75 per cent. of the volatile copper could be caught in a Cottrell treater or a milk of lime absorption tower.

Electrolytic Reduction

The reduction of the copper from cuprous chloride by electrolysis requires only one-half the current which would be required to precipitate it from the cupric form. However, it is difficult mechanically, because the cuprous chloride is a solid and the strongly oxidizing gases given off at the anode must not come in contact with it. This makes necessary the use of a horizontal diaphragm cell and a circulating depolarizing solution. The solution resulting from the SO_2 precipitation after the cuprous chloride settles out is strongly reducing and makes an admirable depolarizing solution, which on oxidation gains the equivalent of about $\frac{3}{4}$ lb. of sulphuric acid per pound of copper reduced. This makes the combination of this method with the SO_2 precipitation method very attractive. The early experiments were made in a cell 6 in. square, shown in plan and section in Fig. 6. It consisted of a wooden box (*a*) with a copper-plate bottom; a tray (*b*) with wooden sides and a sheet-copper bottom; a canvas diaphragm; a graphitized carbon anode; and intake and outlet pipes for the depolarizing solution. The cuprous chloride, filtered and washed, was mixed with about 10 per cent. NaCl, which made it thin enough so it could be readily poured into the tray. The reducing solution was circulated until there was a noticeable smell of chlorine from about the anode. As the run approached completion there was considerable evolution of gas from under the diaphragm. Part of the copper was deposited firmly adhering to the starting sheet and part in a loose form. The data of one of these runs were:

Depth of cuprous chloride, inches	2.0
Average amperage	3.32
Average voltage	2.67
Current density, amperes per square foot	19.14
Theoretical copper deposited, grams	537.4
Copper actually deposited:	
Adhering to starting sheet, grams	225.2
Loose, grams	229.0
Assay of copper:	
Adhering to starting sheet, per cent	99.56
Loose, per cent	97.03
Pure copper deposited, grams	446.4
Current efficiency, per cent	83.06
Power consumption per pound of copper, kw-hr	0.615
Volume depolarizing solution, liters	15.27
Average acidity of head solution, per cent. H_2SO_4	5.23
Average acidity of tail solution, per cent. H_2SO_4	6.97

A combination of three cells similar to those shown in Fig. 7 was then tried. Graphitized carbon immersed electrodes were used, the anode of one cell serving as the cathode of the one above it. Each cell had its own circulating arrangements.

The data of the run were:

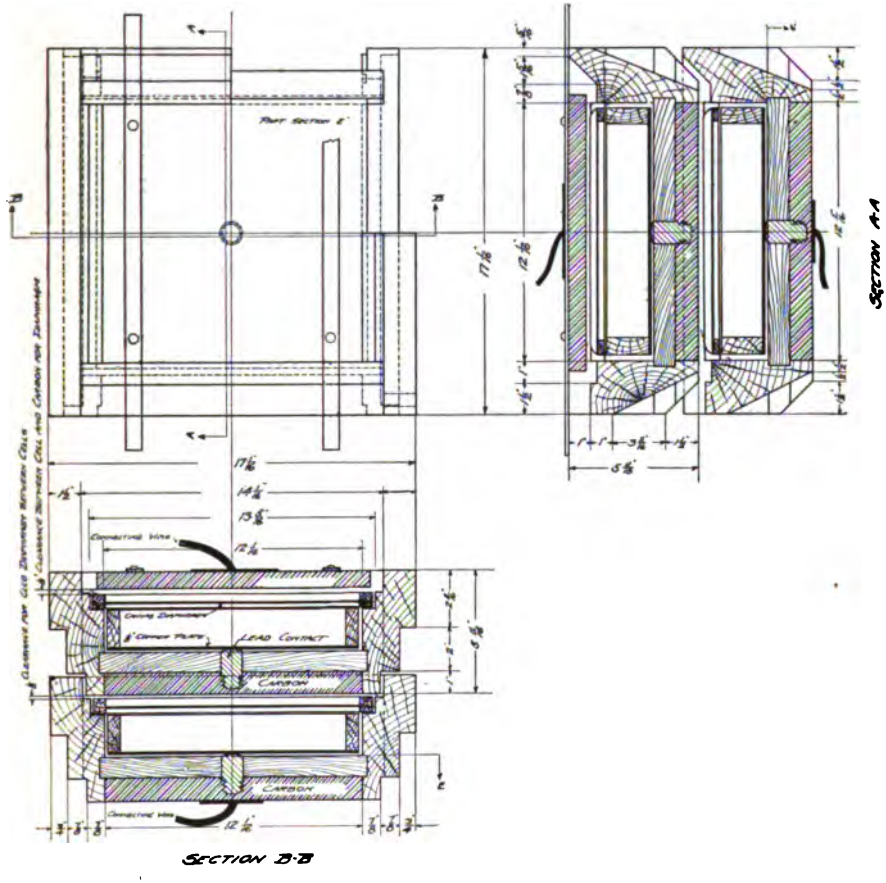

FIG. 7.—PLAN AND SECTION OF TWO ELECTROLYTIC REDUCTION CELLS OF THE 1-FT. SQ. 10-DECK INSTALLATION.

Inside dimensions of trays, inches	1 by 4 by 4
Depth of cuprous chloride, inches	1.0
Total time of passing current, hours	36.16
Average amperage	2.41
Average voltage	7.46
Average voltage per cell	2.49
Current density, amperes per square foot	21.72
Theoretical amount of copper deposited, grams	620.1

Copper deposited:	
Adhering to starting sheet, grams	256.0
Loose, grams	193.7
Assay of copper:	
Adhering to starting sheet, per cent	95.18
Loose, per cent	94.04
Pure copper deposited, grams	425.8
Current efficiency, per cent	68.66
Kw-hr. per pound of copper	0.695
Volume depolarizing solution, liters	18.74
Avg. acidity head depolarizing solution, per cent. H_2SO_4.	4.07
Avg. acidity tail depolarizing solution, per cent. H_2SO_4.	5.64

It was then decided to try the method on a larger scale. Ten cells as shown in Fig. 7 were built and set in a frame so that a cell would slide out from over the one below it, exposing the tray of the lower cell for filling or removal. The cells were set on an incline of $\frac{3}{8}$ in. to the foot, the overflow side being the high side. This made unnecessary the use of a dam on the overflow side to bring the solution in contact with the anode surface.

As yet the cell has not been worked out to an entirely satisfactory basis, but it seems capable of development.

Conclusion

At present it seems advisable to precipitate the copper directly from the copper solutions of the 2,000-ton sands-leaching plant with sponge iron. The precipitation of the copper from such solutions with sponge iron is being worked out and promises to be simple. Until the leaching process is established on a definite operating basis it does not seem advisable to introduce a complicated precipitation method.

The acid plant now building will have ample capacity for the first 2,000-ton leaching unit. When the time comes to build further units, the installation of an SO_2 precipitation plant will depend on considerations of first cost of acid and precipitation plants, cost of operation and other factors which undoubtedly will develop.

The Leaching of Copper Ores

Discussion of the papers of STUART CROASDALE, p. 610; W. L. AUSTIN, pp. 659 and 668; FREDERICK LAIST and HAROLD W. ALDRICH, p. 671; FREDERICK LAIST and F. F. FRICK, p. 691.

R. C. CANBY, Wallingford, Conn. (communication to the Secretary*). —Apropos of the experimental reduction of copper from cuprous chloride by fusion with ground limestone and coke, as described by Messrs. Laist and Frick, and also proposed for use at Chuquicamata, it might be of interest to refer briefly to the experience at Argentine, Kan., in the smelting of the suboxide of copper (as produced in the Hunt and Douglas process) containing appreciable quantities of chlorides.

In the *Engineering and Mining Journal* of June 10, 1911, I gave some account of what had appeared to be crucial difficulties with the Hunt and Douglas process, and how the filter-press difficulty had been so rapidly overcome after my discovery of the effect of the sodium salts in the leaching solutions. I also described the effect of the cupola-furnace gases upon the lime rock with which I had replaced the coke in the absorption tower, which apparently showed the presence of considerable chlorine, due to the imperfect conversion of the subchloride to suboxide, as well as that due to the difficulties of washing thoroughly the suboxide with the means provided, and also to the presence of chlorides in the cement copper which was smelted with the suboxide.

The fumes from the cupola, when issuing direct into the atmosphere, were very dense and white, except when a hot furnace top caused a reddish tinge due to dust. These fumes had a peculiarly suffocating effect, so that when they were blown over into the town the population suffered the greatest actual distress, not the mere annoyance which would have been due to a disagreeable odor. The absorption tower had been built at the instance of the city authorities to abate this nuisance, and not with the idea of making any additional metal savings.

A serious feature in connection with the Argentine experience with the Hunt and Douglas process was, that it had been implicitly assumed that there could be no metal loss in the Hunt and Douglas process itself, and that all attempts to locate the ruinous copper losses which were accumulating in the metallurgical accounts were devoted to the roasting department. It is possibly unfortunate for metallurgical advance along these particular lines that just about the time of this discovery of this previously unsuspected volatilization loss, in the smelting of the suboxide, the Kansas City company made a contract with the Western Union Telegraph Co. for its supply of copper sulphate and the Hunt and Douglas

* Received Aug. 7, 1914.

process was abandoned, the plant being remodeled into a copper sulphate works. I regretted this, as I had confidence that by a more suitable manipulation these volatilization losses could have been overcome, thus giving an ideal treatment for lead-furnace matte, in which the lead, silver, and gold values were so promptly converted into bullion, and in which the copper was equally prompt in being placed in marketable form.

It is obvious that the conditions attending the proposed reduction of the cuprous chlorides by admixture of lime and smelting with coke are very different from those which obtained in the smelting of the suboxide, but the metallurgical experience at Argentine was such as to indicate that in the practical application of the proposed treatment of the cuprous chloride the greatest care must be taken to maintain the proper conditions, otherwise there undoubtedly will be very considerable volatilization losses.

LAWRENCE ADDICKS, Chrome, N. J.—I think this question of whether a process shall be wet or dry is a good deal of a temperance one. Some ten or fifteen years ago there was a good deal of intoxication over wet methods, which was not justified, and the result was that the use of liquor became rather unpopular for a while, and we had a prohibition period. I think now it is largely local option. We have to realize that every one of these cases has to be dealt with as a particular problem, and there is no fixed process that will apply to all cases.

I felt, in looking over these papers, that electrolytic deposition was in need of a champion, and I want to say a few words on that subject. I have been particularly interested in this subject the last three months in doing some work for Phelps, Dodge & Co., and I have been astonished to find under what adverse conditions it was possible to get good deposits. A coherent deposit premises a sufficient number of copper ions in contact with the cathode to satisfy the current. You can get this by having a solution very heavy in copper, or a solution considerably lighter in copper, but with a very rapid circulation. The ordinary electrolytic refiner has a handicap which is entirely absent in leach liquors: namely, the presence of gold and silver slimes at the anode, which cannot be disturbed by any violent circulation, or there will be prohibitive silver and gold losses in the cathode. I believe also that we are wrong in looking at this problem as a cyclical one, rather than a continuous one, and that it would be better to consider the use of liquor quite high in copper, say 3 per cent., take it down to 2 and 2½ per cent., run that back into the tanks and make up the half per cent. that is missing, and then back to the electrolytic tanks. In this way you get better depositing conditions, and, when studied over, it is not so bad from the leaching point of view. Of course, that means counter-current washing of the ore in order to bring the liquor up to the necessary strength

for the depositing tanks, and indicates the use of Dorr thickeners, or something of that kind.

The question of the wash water bringing up the bulk is very easily dealt with in the very dry climates out West, by using a cooling tower and evaporation, to get reduction of bulk.

Now as a depolarizer, using a carbon anode, iron is the real depolarizer in most cases, even though sulphur dioxide may be introduced. Iron is almost always present, and it is a very efficient depolarizer. We found in the experiments at Douglas that it is quite possible to consider working voltages as low as $\frac{6}{10}$ and $\frac{7}{10}$ volt. The depolarizer requires circulation in order to bring fresh iron sufficiently close to the surface of the anode. With such voltages we aim to obtain a recovery of 3 lb. every kilowatt-hour.

There were one or two questions that I wanted to ask, in case anybody could give me any information. One is regarding the apparently remarkable wood out in this country, which seems to hold acid liquors, which wood won't do in the East. I am used, in all my experience in electrolytic work, to see wood after an exposure to sulphuric acid reduced to a soft charcoal, which you can poke your finger through. If redwood will stand acid to the extent which my brief observation indicates, I don't see that there is any problem entailing the use of asphalt mixtures and other protective coatings for this work.

The other question I wanted to put was whether any one here has had experience in the restraining of the solution of impurities by carrying the liquor with which you leach very high in those impurities. The real problem in Arizona is one of alumina, and what to do with it after it gets in solution. My idea was that we might take a solution so high in alumina that the amount dissolved from ore would be very little additional. I might also say we have worked with solutions and electrolytes as high as 7 per cent. alumina, and that we are using in our standard work 3 per cent. Al_2O_3 and 3 per cent. Fe in the liquor without any trouble with the deposit.

STUART CROASDALE, Denver, Colo.—Messrs. Laist and Frick have made an excellent summary of the conditions which govern the selection of a precipitation method in leaching processes for copper ores. The importance of a careful consideration of these conditions is apt to be overlooked by investigators who are over-anxious to develop one idea. The conditions in Anaconda are quite different from those in Chile or even those in Arizona, and a method of precipitation that is adapted to one set of conditions might be quite impracticable in another.

From my own experience, I believe the authors could have obtained a better absorption of hydrogen sulphide by introducing it through a diaphragm of burlap, or of perforated sheet lead like that used for fil-

tration in chlorination barrels, or through a "silica sponge" that was on the market a few years ago for cyanide vats. Any sort of a diaphragm is much better than perforated pipes, provided the column of solution is not too high. With this arrangement, the depth of solution should not be over 6 in., and 3 in. is better. The gas bubbles then have a chance to hit each other and collapse, thus forming a foam with the solution, whereas if they have to go through a greater depth of solution they are forced together into large bubbles and of course escape like so many balloons. A launder, through which a shallow stream of solution could flow over the perforated bottom, would no doubt be a more efficient precipitating apparatus than a tank.

I believe it is possible to make hydrogen sulphide in quantity by a more direct method than by the decomposition of a solid sulphide with an acid. I have produced it quite readily by passing illuminating gas over pyrite at a low red heat, but this probably utilized only one atom of sulphur.

When pyritiferous lignite or bituminous coal is used in a gas producer, considerable hydrogen sulphide is generated.

These facts suggest that both hydrogen sulphide and power gas might be obtained from a gas producer by charging pyrite with the fuel and introducing steam in the proper proportion. A modification of the Hall process for desulphurizing might also yield the desired results.

In making sponge iron it is unfortunate that these gentlemen wasted time and money in duplicating my experiments with the multiple-hearth furnace without making any progress in that direction.

During August, 1912, I made a number of laboratory experiments on the reduction of calcines and iron ore, by using coke, coal, and gas for reducing agents. These were so encouraging that I went East in September, 1912, with Dr. Ricketts and the officials of the Calumet & Arizona Co., to investigate Jones's methods and those that had been used by the various iron companies for making sponge iron.

While visiting the Pennsylvania Salt Works, in Philadelphia, we learned that Professor Crabtree, of the University of Pittsburg, had been experimenting along this line a year or two before with the Wedge muffle furnace, but he had neglected to make his furnace air tight, and, after securing a reduction on the upper hearths, his product oxidized on the lower hearths.

After explaining to Mr. Webb, Secretary of the Furnace company, that we wished to roast pyrite for acid making and then reduce the calcines to sponge iron for a precipitant, we decided to order an experimental furnace of the Wedge double-function type, whereby we could roast the pyrite on the three upper hearths and deliver the hot calcine on the upper of three muffle hearths, where it would be mixed with

powdered coal from a feeder attached to the side of the furnace, and be reduced to sponge iron on the three lower or muffle hearths. This furnace was set up and operated in Douglas, Ariz. We reached the limit of its possibilities, and completed this set of experiments by August, 1913. We learned what not to do and where our furnace was weak. The only thing to do was to build a new furnace along the lines I have already suggested in another paper before the Institute.

It seems that several months later Messrs. Laist and Frick duplicated our furnace in every essential detail, obtained the same results, and stopped their set of experiments where we were forced to stop. Could they have taken advantage of my experience, the time and money spent in duplicating my work would have made substantial progress in making sponge iron from calcines with a continuous furnace. I have gone somewhat into detail on this subject because it emphasizes once more the advantage of more centralized research among companies in the same line of metallurgy.

The days of secrecy have passed, and every metallurgical company in the country keeps open house for the continuous stream of managers, superintendents, and engineers of other companies that go the rounds "swapping information." Some engineers still entertain an imaginary value of patents, but these values have less chance of becoming real as the exchange of information becomes more common. It seems to me, therefore, that under these conditions much greater and more intelligent progress could be made in all branches of metallurgy, whether it be copper, lead, or zinc, if metallurgical companies, or at least closely allied companies, would segregate their research work into common departments in their respective lines and employ a competent force of engineers, metallurgists, electro-chemists, physical chemists, and others, as might be needed to meet the requirements.

This idea has been objected to as being destructive of individuality, subject to the petty jealousies of human nature and subject to the loss of unbiased viewpoints from independent research. It seems to me these are all offset by lack of progress and loss of time and money from the detailed duplication of work which it has been my experience to observe.

To secure cheap iron from calcines, I think efforts should be directed to the development of a continuous furnace. Encouraging results are now being obtained from a shaft type of furnace and some experiments of my own, made in September, 1913, lead me to believe that a cheap and efficient continuous furnace can be made after this pattern.

I do not see how sulphur dioxide can ever become a commercial precipitant for copper from solutions on a large scale. It may be, as the authors state, "highly attractive theoretically," but in leaching large tonnages of low-grade material, as now planned, I do not see how it

can be made practicable to introduce a precipitant into cold lixivium under pressure, then heat the entire lixivium to boiling under pressure, and again cool to ordinary temperature, in order to get a product that has to be treated by another reagent in order to precipitate the copper. I need not mention the other difficulties that have been encountered when using this precipitant in straight copper sulphate solutions. It is, of course, an acid maker, but I believe acid can be made more cheaply by ordinary methods.

In fact, with the average leachable ore, where more or less iron, aluminum, and other elements pass into solution with the copper and the lixiviants soon become foul, I doubt if much advantage can ever be obtained by any method of dual precipitation. The regeneration of the lixiviant is seldom complete, and such a method involves the construction and operation of two distinct plants where cheapness and simplicity of operation are of primary importance.

DORSEY A. LYON, Salt Lake City, Utah.—The following statement appears on p. 660: "When the copper to be extracted is found mineralized as a sulphide the use of sulphuric acid as lixiviant necessitates breaking up the sulphur combinations by roasting in order that the metal can be brought into soluble form. This operation, however, instead of being a drawback, presents advantages. The expense of roasting is light, for it is very effectively and cheaply carried out in modern mechanical furnaces." I would like to inquire from those who have had experience along this line whether they believe this to be true.

FREDERICK LAIST, Anaconda, Mont.—I would like to reply to Mr. Lyon's question, and also to say a few words in reply to Mr. Croasdale's and Mr. Canby's discussion.

I hardly think that any saving of lixiviant due to roasting would balance the cost of roasting in the treatment of low-grade ore. The amount of soluble copper formed during the roasting operation would not cause the saving of much sulphuric acid. It would certainly not save an amount of sulphuric acid which would balance the cost of roasting, particularly when the leaching plant is located at a smelter where acid can be made around $3 per ton.

Regarding Mr. Croasdale's remarks on the absorption of hydrogen sulphide, I am quite certain that a much better absorption than we obtained could be gotten and that a more suitable apparatus than we used could be designed. The question with us, however, was whether hydrogen sulphide could be made cheaply enough from a low-grade copper matte to justify its use as a precipitatant. We never questioned the possibility of getting a complete absorption.

When we undertook to solve the problem of profitably recovering

the copper from our sand tailings by leaching, some three years ago, we decided to do the work thoroughly and to try out every method of precipitation that offered hopes of success. Along with our H_2S experiments we carried on experiments with various other methods, such as precipitation by sponge iron, electrolytic precipitation, sulphur dioxide precipitation, etc. It was quite early in the game that we came to the conclusion that while hydrogen sulphide could undoubtedly be used and developed into a commercial precipitatant, the method would not be as satisfactory as some of the others which were tried. Experimentation, therefore, along this line was dropped and hence there was no incentive on our part for spending the money necessary to develop a more satisfactory type of absorption apparatus than that used in the preliminary work.

Methods for the production of sulphuretted hydrogen, such as passing illuminating gas over pyrite at a low red heat, were tried by us, but did not give sufficiently satisfactory results to compete with the other method. Reactions of this kind are generally very incomplete and for this reason the expense of heating the pyrite and making the illuminating gas is high per unit of H_2S produced.

As regards the making of sponge iron, Mr. Croasdale sees fit to criticise us for wasting time and money in duplication of experiments with the multiple-hearth furnace without making any progress in that direction. Both the making of sponge iron and its use for precipitating copper from solutions are old. Therefore, no one could claim originality for proposing the use of this reagent in connection with the lixiviation work now being carried on by various companies. Professor Lunge describes in considerable detail the making of sponge iron and its use for precipitating copper in his work on sulphuric acid and alkali. The iron oxide which is reduced to sponge iron is cinder or calcine which results from the roasting of Spanish pyrite after the copper has been removed by lixiviation. Soon after we actively commenced experimentation here it occurred to us that the iron which is so abundant in our calcines might be converted into a metallic form and used for precipitating copper. We looked up the subject of sponge iron and could find records of no apparatus devised for making it which fully met our requirements.

During the Butte meeting of the Institute, in the course of conversation with W. McA. Johnson, he suggested the use of a muffle-fired MacDougall furnace. At that time, we had not arrived at any clear idea as to what form of reduction furnace could be used on a large scale, and we were building a small retort furnace in order to obtain several hundred pounds of sponge iron for experimentation. Mr. Johnson's proposal sounded pretty good to us and we decided to fix up one of our MacDougall furnaces and give the method a trial. We did not know at that time exactly what work Mr. Croasdale was engaged on, nor

would such knowledge have materially influenced our actions. The cost of making the experiment was not great, and while it was an absolute failure, it clearly showed us wherein lay the defects of the MacDougall furnace for this kind of work and served to put us on the right track.

The MacDougall furnace is admirably adapted for oxidizing, and for that very reason it is not adapted for carrying on operations where a powerfully reducing atmosphere must be maintained. It is almost impossible to entirely exclude the air, and it is impossible to heat the charge to the reduction temperature by means of muffles without the consumption of an excessive amount of fuel. Moreover, the expense of keeping the muffles in repair is prohibitive.

We, therefore, turned our attention to the Brueckner cylinder as a more promising type of furnace for this work. We rapidly carried out a series of experiments, starting with a small hand-turned cylinder, and finally ending up with the present 8½ by 12 ft. cylinder, in which we are making about 20 tons of sponge iron per day, with a moderate consumption of fuel and the production of a high grade of sponge iron.

Mr. Croasdale further criticises the use of SO_2 as a precipitant and thinks the expense of working under pressure, heating the solutions, etc., must be very formidable obstacles. We naturally investigated these phases quite thoroughly before we did much work along this line. The pressures required are moderate and involve no more difficulties than blowing a converter charge; in fact, the pressure is the same (16 lb.) as is used in converter practice. The saturating vessel need not be closed provided it is made sufficiently high so that the lower portions of the liquid will be subjected to this pressure. The most expensive part of the operation is heating the solutions. This expense can be materially lessened by the use of counter-current apparatus in which the hot precipitated solutions are made to heat the cold solutions from the absorber. Our calculations indicate that precipitation by means of SO_2 can be accomplished for around $10 per ton of copper, without taking any credit for the regenerated acid.

I am strongly of the opinion that where electrolysis cannot be used, owing either to high cost of power, rapid fouling of the solutions, or the necessity for maintaining chlorides in the solutions owing to the presence of silver in the ore, precipitation by SO_2 is not only the best way out of the difficulty but is an entirely practical method.

As Mr. Canby states, the reduction of copper from cuprous chloride is accompanied by a certain amount of volatilization. This is more marked when cuprous chloride is first decomposed by milk of lime, as was done in the Hunt-Douglas process, the resulting cuprous oxide, so called, being smelted. We found in our experiments that the prod-

uct obtained in this way was a mixture of oxychloride and suboxide of copper. When you try to smelt this the volatilization losses are very heavy. The method that we used, and which is also proposed by Mr. Smith for Chuquicamata, does not give the copper much chance to volatilize because the limestone present undergoes double decomposition with the cuprous chloride long before the volatilizing point is reached. Our experiments indicate that about 5 per cent. of the copper is volatilized, 75 per cent. of which could be recovered from the fume, so that the net loss of copper would be quite small.

W. L. AUSTIN, Riverside, Cal. (communication to the Secretary*).—Replying to the question raised by Mr. Lyon: If a sulphide ore is to be leached with sulphuric acid solution it must of necessity be previously oxidized, and roasting will usually prove to be the most economical method of oxidation. Roasting puts the ore in excellent condition for leaching. For example, a slime that cannot be leached in the state in which it is found in a tailing pit, can be readily treated after roasting. Roasting furthermore sulphatizes lime present, and converts the iron into more or less insoluble ferric oxide. Sulphur dioxide used in the electrolytic vats as a depolarizer can be obtained from roaster gases. Roaster gases when used in this way may furnish all the acid required for leaching, and may considerably reduce the amount of current consumed in electrolytic deposition. For these reasons, roasting, "instead of being a drawback, presents advantages" in many instances; but of course an ore both economically and metallurgically suitable for the purpose indicated is presupposed. The author had no intention of advocating roasting all low-grade ore with a view to saving lixiviant by production of sulphates. Ore which is amenable to leaching is unfortunately seldom found where sulphuric acid is obtainable "around $3 per ton."

† On p. 636 Mr. Croasdale's paper, Leaching Experiments on the Ajo Ores, a table is given showing the amount of acid consumed when leaching said ore by percolation. Analyses made of the liquors during the second and third applications of lixiviant indicate approximately the combinations entered into. The following figures are taken from the tables mentioned:

First and last analyses; second lixiviant:

	H_2SO_4, Per Cent.	Cu, Per Cent.	Al_2O_3, Per Cent.	Fe (Ferric), Per Cent.	Fe (Ferrous), Per Cent.
Second lixiviant.....	10.31	0.9	0.57	0.49	0.13
9 p.m.....	0.72	3.0	0.94	0.40	0.49
	− 9.59	+ 2.1	+ 0.37	− 0.09	+ 0.36

* Received Sept. 18, 1914.
† Received Sept. 25, 1914.

Assuming that these several metals were present in the lixivium as sulphates, the following quantities of acid may be accounted for:

	Per Cent.
H_2SO_4 taken up by the copper	3.24
H_2SO_4 taken up by the Al_2O_3	1.06
H_2SO_4 taken up by the Fe (ferrous)	0.63
	4.93
H_2SO_4 released by the Fe (ferric)	0.24
Total acid consumption accounted for	4.69
Free acid unaccounted for	4.90

With the third application of lixiviant the account runs as follows:

	H_2SO_4, Per Cent.	Cu, Per Cent.	Al_2O_3, Per Cent.	Fe (Ferric), Per Cent.	Fe (Ferrous), Per Cent.
Third lixiviant	10.00	0.4	(0.58)	0.16	0.26
9 p.m.	1.44	2.54	0.98	0.55	0.54
	− 8.56	+ 2.14	+ 0.4	+ 0.39	+ 0.28

	Per Cent.
H_2SO_4 taken up by the copper	3.3
H_2SO_4 taken up by the Al_2O_3	1.15
H_2SO_4 taken up by the Fe (ferrous)	0.49
H_2SO_4 taken up by the Fe (ferric)	1.02
Total acid consumption accounted for	5.96
Free acid unaccounted for	2.6

In the analysis given on p. 613 the oxidized ore is shown to contain 0.56 per cent. CaO, but it is reasonable to assume that this was neutralized by the first lixiviant, and as apparently the three lixiviations were carried out on one charge, the small amount of CaO present in the ore could not have influenced acid consumption in the second and third lixiviants. There are no other elements mentioned in the analyses which might account for the excess acid consumption. As free acid in large quantities was removed from the lixiviant, it must have gone into combination with something, and consumption of acid has an important economic bearing on leaching porphyritic ore with sulphuric acid. On p. 641 it is stated that 3.6 lb. H_2SO_4 (not commercial acid) was consumed on the average per pound of copper dissolved.

Furthermore, it is shown in the diagram on p. 637 that in the initial stages of the treatment free acid is used up very rapidly, out of all proportion to the copper going into solution. It would add to the value of Mr. Croasdale's excellent paper if he would throw further light upon the points raised.

In general, when a comparatively weak sulphuric acid lixiviant is used, the copper in oxidized ore is for the greater part removed before the other elements are seriously affected, whereas a strong (10 per cent. acid) lixiviant attacks all bases simultaneously. The objection made by Mr.

Croasdale to the use of a weak lixiviant—namely, that neutralizing alkalinity in the ore needlessly consumes time—might be overcome by making the lixiviant stronger in the beginning, and then carrying on the operation with a weaker solution.

The writer has found in some cases that by reducing an oxidized ore to 16 mesh, and violently agitating with a strong acid solution, the copper can be dissolved in a surprisingly short time without undesirable quantities of other elements appearing in the lixivium. In this way it is sometimes possible to dissolve the metal with an acid consumption of less than 1 lb. H_2SO_4 per pound of copper extracted.

Some tests made to determine the effects of lixiviants of varying acidity in dissolving copper from oxidized ore showed that with a certain percentage of acid the lixiviant acted practically on the copper alone. When a certain point was passed the acid was rapidly consumed without correspondingly effective action on the copper.

In connection with acid consumption at Ajo, it would be interesting to know what proportion was neutralized by CaO in the concrete of which the vats were constructed. Concrete will stand a cupriferous solution carrying 1 or 2 per cent. H_2SO_4 fairly well, but it is rapidly eaten into when 10 per cent. solutions are used.

STUART CROASDALE.—The table on p. 636 referred to by Mr. Austin was prepared only for the relative value of the results shown.

In my leaching operations it was impracticable to remove each lixiviant completely before adding another, or even to allow a space between the two, on account of the danger of entrapping some air, which would have a tendency to produce channels or imperfect leaching after the ore had been once saturated. Therefore, as soon as the first lixiviant was drawn down to the top of the charge, the second lixiviant was applied. I then continued to draw off the first lixiviant until its estimated volume was removed, before I began the circulation of the second lixiviant, but there was always an indefinite and considerable amount of the first lixiviant retained by the ore, which became mixed with the second lixiviant as soon as circulation began. The same would be true of the second and third lixiviants, so it would be impossible to account for the acid consumption from these percentages alone, as Mr. Austin has figured it.

The acid consumption per pound of copper dissolved, as given on p. 641, was found by determining the percentage of acid, the specific gravity, and the volume of all solutions and wash waters. It was also estimated by determining the combined acid from the analyses of these solutions at the end of a test or clean up from 75 to 100 or more tons of ore.

As shown, these results are 3.6 and 3.15 respectively, which is a very close check, considering the slight but unavoidable errors in making free-acid determinations in solutions containing salts of iron, aluminum, and copper.

Melting of Cathode Copper in the Electric Furnace*

BY DORSEY A. LYON AND ROBERT M. KEENEY, PITTSBURG, PA.

(Salt Lake Meeting, August, 1914)

INTRODUCTION

THE electric furnace has always been found to be especially adapted to melting, refining, and finishing processes throughout its gradual acceptance by metallurgists as a practical apparatus for conducting metallurgical operations. In the steel industry, the electric furnace is firmly established in the manufacture of steel of the highest grade, equal to crucible steel. For the production of the cheaper grades of steel in large tonnage, the electric furnace gives a higher-grade product than the open-hearth or converter, but due to the high cost of power prevailing in steel centers, considered from the electric furnace standpoint, it is not economical to produce tonnage steel in the electric furnace. As a refining and finishing agent for open-hearth or Bessemer steel, the electric furnace has had some degree of success in the production of tonnage steel.

After a study of the use of the electric furnace in the steel industry, it appears that there may be a possibility of its use to advantage for the melting of cathode copper. In the case of copper the problem is not one of actual refining, because the copper if refined properly by the electrolytic method needs no further refining; it needs simply to be cast into a marketable shape. The name "refining" is applied to the present finishing process, because in the operation oxygen and other impurities are removed, which the electrolytic copper absorbs in being melted in the reverberatory furnace. Considering that processes may be positive, negative, and neutral, the electrolytic method would be positive; reverberatory melting of cathode copper, negative; and electric furnace melting, neutral. The electrolytic method produces about as pure a copper as is possible, if operated properly and the anodes are not too impure. The final reverberatory melting of this cathode copper lowers the grade of the final product, due to the absorption of gases and impurities.

The reverberatory melting of cathode copper is one of the weak parts

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of the metallurgy of copper to-day. A refining process is not required, for before the copper is melted in the reverberatory furnace it is pure. The ideal process would be one in which the cathode copper could be melted and cast into marketable shapes without absorbing impurities; *i.e.*, a neutral process.

Reverberatory-Furnace Refining

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[1] Burns, W. T.: Notes on the Great Falls Electrolytic Plant, *Trans.*, xlvi, 703 (1913)

That the furnace refining of copper is superfluous in so far as it improves the grade of marketable copper has been shown by the experiments of E. Keller.[2]

As a result of these experiments Keller states that "electrolytic copper which has not been melted and refined is superior in conductivity to the refined and cast copper by over 2 per cent. There is, therefore, a field for improvement in the refining or making of wire from electrolytic copper without previous melting." Peters states[3] that "ordinary electrically deposited copper produced on a large scale at a plant using the Emerson system of depositing sheets direct without subsequent furnace refining showed a conductivity of from 102 to 103 per cent."

Another test[4] showing the furnace-refining operation to be superfluous, or even harmful, was made by rolling wire from a mass of Lake Superior native copper and, after annealing it, comparing it with cathode copper and furnace-refined native copper. The mass copper and cathode copper each had a conductivity of 102.5 per cent., while furnace-refined native copper had a conductivity of 99.5 to 100 per cent.

Effect of Cuprous Oxide on Copper

As before stated, in the reverberatory process, while the copper is being melted, it absorbs oxygen, sulphur and other impurities from the gas passing over it from the fire box of the furnace; when air is blown into the molten copper, about 5 to 6 per cent. cuprous oxide is found in the charge. This cuprous oxide gives up its oxygen to the metallic impurities in the copper, forming oxides, which pass into the slag. After the removal of all the sulphur dioxide gas the cuprous oxide is reduced to metal by poling until the button sample shows the proper structure and the copper is tough pitch, when it is cast. The cathode copper contains less than 0.1 per cent. cuprous oxide. After furnace refining it contains 0.4 to 1.2 per cent. cuprous oxide, with an average of about 0.7 per cent.

It seems that under the present conditions of refining, cuprous oxide is essential to the quality of the copper produced, as is shown by the fact that if the poling is continued below certain limits the quality of the metal becomes impaired.

It does not appear, however, that the presence of cuprous oxide itself has the direct effect of improving the quality of the metal, as mass Lake copper without previous melting shows a higher all around standard than furnace-refined Lake copper. Hampe[5] found that the addition of cuprous oxide produced no perceptible effect upon the strength or malleability

[2] Keller, E.: *Mineral Industry*, vol. vii (1898); Peters, E. D.: *Principles of Copper Smelting*, p. 484.

[3] *Principles of Copper Smelting*, p. 484.

[4] *Idem*, p. 485.

[5] *Zeitschrift für Berg-Hütten und Salinenwesen* (1873, 1874, 1876); Peters, E. D.: *Principles of Copper Smelting*, p. 491.

of pure copper until 0.45 per cent. was added, when a slight diminution of tenacity was observable. With increasing proportions of cuprous oxide, the quality of the copper suffered more perceptibly.

Addicks[6] found that the addition of 0.05 per cent. oxygen (0.44 per cent. Cu_2O) to pure copper increases slightly its conductivity, which drops back again to about normal when the addition of oxygen reaches 0.1 per cent. (0.9 per cent. Cu_2O) and decreases considerably by further addition.

The theory has been advanced that the presence of cuprous oxide improves the quality of the copper by preventing the reduction of the dissolved foreign oxides, harmless while oxidized but injurious when reduced to the metallic state. These impurities may also be reduced during over-poling to reduce the cuprous oxide.

Peters states, "We may, I think, say with safety that, while the proportion of cuprous oxide found in ordinary good refined copper does not appear to diminish its electrical conductivity (it may even increase it slightly), the very highest conductivity tests are yielded by the copper which contains no determinable oxygen and that cuprous oxide, in the proportion usually found in good refined copper, appears to have but little effect, one way or the other, upon the malleability, ductility, or electrical conductivity of the metal."

Conditions Necessary for the Electric-Furnace Melting of Cathode Copper

From the preceding discussion of present practice in melting cathode copper, it can be seen that the ideal process for melting cathodes is a method by which it is possible to produce cheaply in marketable form as pure a copper as is charged into the melting furnace. Such a process to be successful both metallurgically and commercially should conform to the following conditions:

1. In order to keep the percentage of impurities as low in the final product as in the cathode copper charged into the furnace, there should be a neutral atmosphere in the melting chamber with a minimum possibility of introduction of air, gases, or impurities which might contaminate the copper.

2. Melting should be performed without excessive losses of copper, either by volatilization or in any slag which cannot be returned to the blast furnace or reverberatory furnace for resmelting.

3. The process should be susceptible to ready mechanical manipulation for charging and tapping.

4. The cost of production should be at least as low as by existing methods, which means that the furnace must handle a large quantity of copper per day with an efficient use of labor, fuel, and power.

[6] *Trans.*, xxxvi, 18 (1905); Peters, E. D.: *Principles of Copper Smelting*, p. 491.

The reverberatory process possesses all of these qualifications with the exception of the first, for, as has been shown, air, gases, and impurities are absorbed by the copper and must be removed as far as possible later in the operation. It is not simply a neutral melting furnace. It melts copper cheaply, is easily charged and tapped, and has no apparent loss by volatilization, but produces a poorer grade of copper than it receives.

Factors Governing Use of the Electric Furnace

The use of the electric furnace for melting cathode copper would depend largely upon the cost of operation and the loss of copper by volatilization. The cost of operation would be influenced chiefly by the cost of hydro-electric power in comparison with the cost of coal, with the higher efficiency of the electric furnace in its favor. A loss by volatilization has been found to occur by experimenters using the direct-arc type of furnace with the copper covered with slag. There has been no loss in this manner with the indirect-arc type of furnace, heating by radiation of the arc, or in the resistance furnace.

This is a question requiring further experimenting. A large electric furnace is easily manipulated both for charging or discharging, as has been found to be the case with electric steel furnaces, and lends itself readily to the use of mechanical devices for charging cathodes and casting the copper.

Advantages Offered by the Electric Furnace

Disregarding the commercial economy of an electric furnace for melting copper, the electric furnace provides an absolutely neutral melting chamber with no possible introduction of air or impure gases of combustion during the melting of the copper. Hence, since electrolytically deposited copper is practically pure, the copper melted in the electric furnace should be as pure as cathode copper, and subsequent oxidation and poling are not necessary to remove impurities acquired in melting, as none are acquired. That a product entirely free from oxygen can be produced by melting copper in the electric furnace has been shown, as stated in a previous paper by the writers.[7]

Native copper concentrates which contained about 35 per cent. copper were melted in an arc furnace, the slag being formed by the gangue of the concentrate with a suitable flux. As a result of 17 tappings of the furnace, copper was produced in which no trace of oxygen could be found after etching with picric acid and studying the results with the microscope. To confirm these results, the same samples were submitted to E. S. Bardwell, of Great Falls, Mont., for etching with hydrogen and

[7] The Smelting of Copper Ores in the Electric Furnace, *Trans.*, xlvii, 233 (1913).

determination of oxygen by his method.[8] After an examination of 12 samples Mr. Bardwell confirms the results of the writers by reporting that he can discover no oxygen in the samples after etching with hydrogen.

Types of Electric Furnaces that Might be Used for Melting of Cathode Copper

There are four types of furnaces which might be used for the melting of cathode copper: (1) the induction furnace; (2) the resistance furnace; (3) the direct-arc furnace; and (4) the indirect-arc furnace.

Of these furnaces the *induction furnace* is not practical, for the following reasons: (1) The resistivity of the copper is so low that the current necessary to produce the required heat must be so great that the pinch effect severs the column of molten metal; (2) the induction furnace is not adapted to melting cold metal; and (3) the size and shape of the hearth would make it difficult to charge large quantities of cathodes. It is stated by one authority that when copper is melted in the induction furnace, the column breaks off so often that it is necessary to have a crucible of molten metal ready to pour into the break.

The *crucible resistance furnace* with a solid resistor is too small for melting large quantities of copper, and as now constructed is not practical in large sizes. A resistance furnace of the type used by FitzGerald,[9] with a conducting roof which acts as a resistor and from which heat is reflected to the metal beneath, would not be practical in very large sizes because of the weakness of the roof. Also this type of furnace was found to have a low thermal efficiency.

Electric furnaces of the direct resistance type in which the molten bath is the resistor have not been successful in the steel industry, and attempts to use this principle have been abandoned. The recent use of the "*pinch phenomenon*" in resistance furnaces with a molten resistor, however, has been successful in small installations. Even in melting steel in such a furnace it is necessary to use a very low voltage to get the high amperes necessary for heating. The transformers are attached to the bottom of the furnace, which, while all right in small sizes, might cause complications in a large furnace. With copper in the furnace an even lower voltage would be necessary, adding to the difficulties of design. For molten steel a voltage of from 6 to 10 volts has been necessary.

The pinch furnace does not seem adapted to a large capacity per charge, and as yet has not been built in sizes approaching arc steel furnaces. While it might be used for melting about 500 lb. of cathode copper per charge, with short successive runs, this scheme does not seem prac-

[8] *Trans.*, xlvi, 742 (1913).

[9] FitzGerald, F. A. J.: *Transactions of the American Electrochemical Society*, vol. xix, p. 273 (1911); vol. xx, p. 281 (1911).

tical for a plant melting 100 tons of copper per day. When copper was tapped at intervals of about 10 min., it would be difficult to use large-scale charging and casting machinery, which reduces the labor cost very much. The wear on the lining of the furnace would be greater than when a large furnace is used and discharged at intervals of several hours.

Another disadvantage of such a furnace is that in reverberatory melting large amounts of copper are absorbed by the lining, and this would doubtless be the case also in electric-furnace melting. This would prove a great disadvantage to the use of an electric furnace with a conducting hearth of any sort. While the hearth of the pinch furnace is not conducting over its entire cross-section, it is conducting at the points where the injectors or electrodes are placed in the hearth. After absorption of copper by the hearth short circuits might occur between the electrodes, and if the furnace were relined before affairs could reach this point, it would be a case of relining the furnace very often, at great expense. In the case of a furnace with a non-conducting hearth, the copper could be permitted to accumulate on the hearth without injuring the operation of the furnace.

In regard to the use of the pinch furnace for melting cathode copper, there is no question that it gives the best metallurgical conditions for carrying on the operation in a neutral atmosphere, but, in the opinion of the writers, it does not seem, from present developments, to be certain of success for this purpose in a practical way on a large scale. There are too many problems to be worked out in regard to the practical application of this type of furnace to warrant its use for tonnage melting at the present time. It provides a neutral atmosphere, with no chance of the introduction of impurities, and has one advantage over the direct-arc type of furnace in that there is not the necessity of keeping a bath of slag over the copper to prevent loss by volatilization. It is practically impossible for any volatilization of copper to occur in a pinch furnace.

Another possible difficulty in the use of the pinch furnace on a large scale is the necessity of keeping a molten bath of copper in the furnace in order to operate it. In charging, cold solid copper would be placed into hot molten copper. If there was any splashing of the copper, there might be a tendency for copper to collect on the roof, as the roof is the cold part of the furnace, since all the heat is generated in the metal charged.

The *indirect-arc furnace* in which an arc is drawn between electrodes, and in which the passage of the current is entirely independent of the molten bath, has the advantage of having the heating independent of the slag or metal. In steel melting the very basic slag used tends to promote the steadiness of the arc. Hansen[10] states that the arc in a zinc furnace or brass-melting furnace of this type is rather snappy and unsteady. This

[10] Hansen, C. A.: *Transactions of the American Institute of Metals*, vol. vi, p. 110 (1912).

is apparently not fatal, for in his experiments with the Weeks furnace the arc was operated for 42 hr. continuously in an atmosphere of zinc vapor. The electrodes were not adjusted once during this time.

For copper melting on a large scale a furnace with from three to six electrodes probably would be used. Under such circumstances it would be necessary to have the electrodes controlled hydraulically by hand, as in the indirect-arc steel furnace, or by automatic electric control. From experience with the indirect-arc steel furnace it is evident that with furnaces of this type difficulty occurs in the regulation of electrodes and the regulation of the arc for a capacity of over 2 tons per charge. The larger size seems to present complications too great to permit of satisfactory operation. The 5-ton furnace built at Turin failed largely for this reason, and in steel melting it has been evident that the indirect-arc furnace is not adaptable in sizes of over 2 tons, or 300 kw. power load. In his first trials with the electric furnace, Stassano built a rectangular hearth furnace with six horizontal electrodes arranged in three pairs, lengthwise of the furnace. This furnace failed largely because of the difficulty in maintaining the arc and the lining.

One of the greatest objections to the indirect-arc furnace for steel manufacture has been the high repair cost for linings resulting from the direct radiation of the heat of the arc. Magnesite linings, which increase the expense considerably, are necessary. The heat losses in water-cooling the electrodes are also larger than in some other types of electric steel furnaces.

As a result of experience with the indirect-arc steel furnace, while there would be less chance of loss of copper by volatilization than in the direct-arc type, it is believed that the furnace would be too complicated for practical work; the up-keep cost too expensive; and the heat losses high. This furnace, while giving the proper atmosphere for carrying on the melting of copper, does not seem to be a practical, efficient furnace for large-scale work.

The *direct-arc furnace*, in which the arc is formed directly between the electrode and the slag or metal, may be of the conducting or non-conducting hearth type. The conducting hearth type is not feasible for copper melting, as before stated, because it has been found that the combustion reverberatory furnace used in copper melting absorbs large quantities of metallic copper. Also there is always the danger of metal breaking through the bottom of a conducting hearth, which would be more apt to occur in the case of copper than steel, on account of the lower melting point and greater fluidity of the metal.

After a careful study of the existing types of electric furnaces, it is believed that the most practical furnace for copper melting would be the direct-arc type, having a non-conducting hearth, as for example the Heroult steel furnace. As an efficient, practical furnace, this furnace has

demonstrated its superiority in the metallurgy of steel. The objections to the direct-arc type as not affording theoretically the best possible conditions for copper melting do not, in the opinion of the writers, overweigh the practical points in its favor.

There are three chief disadvantages to the direct-arc electric furnace for the melting of cathode copper:

1. In order to prevent volatilization it is necessary to keep slag on the bath at all times when current is passing through the furnace.

2. From experimental work to date, it appears that there is a slight volatilization of copper with a furnace of this type even when a slag is used.

3. The presence of slag makes it difficult to discharge the furnace and keep the copper at the proper temperature for pouring.

With the exception of loss by volatilization, these objections do not seem serious, and the volatilization loss should be very small if the furnace is operated largely as a resistance furnace with a thick enough bed of slag (2 to 3 in.) to prevent direct arcing with the metal, and a low current density in the electrode to reduce excessive local heating at the points where the electrodes dip into the slag. Hansen states that in a newly lined furnace not designed for copper melting he tapped 99 per cent. of the metal charged. The actual loss could be determined only after a campaign of several months. If loss occurred, it would probably be at the start of a run when the slag is cold and not fused.

The question of discharging the furnace is a problem of operation. The copper could be cast by hand from a ladle, as is done at Great Falls. In this case it would be necessary to preheat the ladle and skim the slag from it before pouring, if the slot tap hole usually used on reverberatory melting furnaces is employed. Or a preheated bottom-casting ladle could be used, as in making steel castings. A siphon tap hole might be designed so that the copper drawn came from the bottom of the bath, leaving the slag on top of the metal. This tap hole would be lowered gradually, as in present melting practice, taking the metal from the bottom, but with only the pressure of the surface of the bath behind it. Only the last of the copper drawn would contain slag, and the current could be maintained on the furnace enough to keep the copper at the proper casting temperature until the slag was lowered to the bottom of the furnace. Thus bottom casting, if preferable to skimming in the ladle, would not be necessary except at the end of the operation.

For machine casting, with a siphon tap hole the operation would be performed as in reverberatory practice, except that it would be necessary to skim the last ladles or use a bottom-casting ladle on account of the slag. It is simply a question of manipulation.

In the electric process of melting cathodes with the direct-arc furnace, the charge would be placed in the furnace mechanically with a charging

machine, as in reverberatory practice. The electrodes would be drawn up during charging. After the cathodes were all added to the furnace, lime and silica would be shoveled in to form a protective covering of slag. The slag should be of as low a melting point as possible. The electrodes would then be lowered, the furnace shut up and the circuit closed. The electrodes would be regulated by hand at first and later by Thury regulators, as in steel furnaces. Under these circumstances, as the slag is practically pure and no air is admitted into the furnace, poling is not necessary and the furnace can be tapped when the charge is all melted and at the proper casting temperature. During tapping the electrodes would be gradually lowered and the current kept on, so as to maintain the proper temperature. When operated with a slag 2 to 3 in. thick, the furnace is really a resistance furnace rather than a direct-arc furnace.

Because of the small amount of work done with melting of copper in the electric furnace, accurate estimates of the cost of the process are difficult to obtain. It is probable that the power consumption would not be over 300 kw-hr. per ton of copper melted. Using this figure and basing other expenses on electric-furnace steel practice, the cost of melting cathode copper in a furnace of 25 tons capacity per charge should not exceed $4.75 per ton, or 0.238c. per pound, of copper melted, with power at ½c. per kilowatt-hour.

Conclusion

This short discussion of the subject is not offered with the idea of attempting to convert those interested in the matter to the belief that the only thing to do is to melt their copper in an electric furnace, but rather to suggest that the electric furnace might be used for that purpose.

We also hope that the paper may be the means of starting a discussion of the subject, by melters of copper and electric furnace men alike, for, as is well known, quite often it is not the paper itself, but the discussion of it, that makes it valuable, and so we hope it will be in this case.

Discussion

Lawrence Addicks, Chrome, N. J.—I read this paper for the first time this afternoon, and it is really worthy of a much more careful reply than I can make with such a short time to consider it, but I just want to say that I differ with Dr. Lyon in his premises and conception of what happens in a refining furnace. He cites steel as an instance where the production of special grades justifies the high cost of power in an electric furnace, and goes on to say the same will be true in the case of copper, for the reason that you will get a better quality of product. I don't think you will get a higher grade of copper, or a better conductivity, as I will explain in a moment. The cost of fuel in the Eastern

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BY DORSEY A. LYON AND ROBERT M. KEENEY, PITTSBURG, PA.

(Salt Lake Meeting, August, 1914)

INTRODUCTION

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After a study of the use of the electric furnace in the steel industry, it appears that there may be a possibility of its use to advantage for the melting of cathode copper. In the case of copper the problem is not one of actual refining, because the copper if refined properly by the electrolytic method needs no further refining; it needs simply to be cast into a marketable shape. The name "refining" is applied to the present finishing process, because in the operation oxygen and other impurities are removed, which the electrolytic copper absorbs in being melted in the reverberatory furnace. Considering that processes may be positive, negative, and neutral, the electrolytic method would be positive; reverberatory melting of cathode copper, negative; and electric furnace melting, neutral. The electrolytic method produces about as pure a copper as is possible, if operated properly and the anodes are not too impure. The final reverberatory melting of this cathode copper lowers the grade of the final product, due to the absorption of gases and impurities.

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INTRODUCTION

THE electric furnace has always been found to be especially adapted to melting, refining, and finishing processes throughout its gradual acceptance by metallurgists as a practical apparatus for conducting metallurgical operations. In the steel industry, the electric furnace is firmly established in the manufacture of steel of the highest grade, equal to crucible steel. For the production of the cheaper grades of steel in large tonnage, the electric furnace gives a higher-grade product than the open-hearth or converter, but due to the high cost of power prevailing in steel centers, considered from the electric furnace standpoint, it is not economical to produce tonnage steel in the electric furnace. As a refining and finishing agent for open-hearth or Bessemer steel, the electric furnace has had some degree of success in the production of tonnage steel.

After a study of the use of the electric furnace in the steel industry, it appears that there may be a possibility of its use to advantage for the melting of cathode copper. In the case of copper the problem is not one of actual refining, because the copper if refined properly by the electrolytic method needs no further refining; it needs simply to be cast into a marketable shape. The name "refining" is applied to the present finishing process, because in the operation oxygen and other impurities are removed, which the electrolytic copper absorbs in being melted in the reverberatory furnace. Considering that processes may be positive, negative, and neutral, the electrolytic method would be positive; reverberatory melting of cathode copper, negative; and electric furnace melting, neutral. The electrolytic method produces about as pure a copper as is possible, if operated properly and the anodes are not too impure. The final reverberatory melting of this cathode copper lowers the grade of the final product, due to the absorption of gases and impurities.

The reverberatory melting of cathode copper is one of the weak parts

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of the metallurgy of copper to-day. A refining process is not required, for before the copper is melted in the reverberatory furnace it is pure. The ideal process would be one in which the cathode copper could be melted and cast into marketable shapes without absorbing impurities; *i.e.*, a neutral process.

Reverberatory-Furnace Refining

As cathode copper is practically pure copper, it would be ready for use if in a marketable shape, but as it is not, the cathodes must be melted. Common practice is to melt cathodes in the reverberatory furnace. The practice at Great Falls, Mont., has been described by W. T. Burns[1] as follows:

Two coal-fired reverberatory furnaces of 100,000 lb. capacity each per charge are used for melting cathodes. The coal used in firing them contains about 3.5 per cent. sulphur. On account of this high sulphur content in the coal, it is necessary to protect the copper as it melts from the sulphur dioxide gases resulting from combustion of the fuel. In order to do so about 30 per cent. of the cathodes going to make up a wire-bar charge are dipped in milk of lime before charging. As the copper melts the lime forms a protective coating for the metal, thus hindering in a large measure the absorption of sulphur by the molten copper. Just before rabbling (or oxidizing) is started the slag is skimmed from the furnace. The slag obtained is equivalent in weight to about 3.5 per cent. of the copper charged, varying with the amount of lime added. The slag contains about 60 per cent. copper.

After the charge is completely melted, the usual practice is followed of rabbling and poling the copper for oxidation of the impurities and bringing the metal to the proper pitch for casting. Rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath. The poling and reducing action is obtained by forcing green pine poles into the bath and by covering the surface of the bath with charcoal.

The condition of the bath, as regards oxygen content, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the "tough pitch" stage has been reached, the poles are removed and the dipping operation is begun.

Why Reverberatory Refining is Superfluous

The process of reverberatory melting of cathode copper as above outlined is not a refining process. That it is so called is due to the fact that it is necessary to remove the oxygen and other impurities which the electrolytic copper absorbs during melting in the reverberatory furnace

[1] Burns, W. T.: Notes on the Great Falls Electrolytic Plant, *Trans.*, xlvi, 703 (1913)

in the fire; that is, it gave off its volatile matter so rapidly that there was a good flame in the furnace for a few minutes, but for most of the time there was practically no flame. To minimize this effect, the fire-box temperature was kept as low as possible, by letting a thick bed of ash accumulate. This procedure also prevented much loss of fuel through the grate; but, as the ash was fine and did not clinker, the grating had to be done cautiously. A furnace would use about 25 per cent. more of Rock Springs than of Diamondville coal, but would not smelt more than 90 per cent. as much material.

A number of trials have been made at the new works with mine-run coal from Bear Creek, Mont. The first shipment was received before the mines were much developed; and the natural-draft results were like those with Belt. (The Belt mines, however, were well developed). With the coal 24 in. deep in the 7 by 16 ft. fire box the furnace froze in 6 or 8 hr., despite the best efforts with different methods of firing, and while using coal at the rate of 72 tons in 24 hr. The cause of this failure was lack of flame. This same shipment of coal succeeded, however, under forced draft, even though in one case it was used at a slower rate than during the natural-draft trial; and this was due to the intense combustion on the grate distilling some heavy volatile matter into the hearth, and developing flame there. This is parallel to the experience with Belt coal. Having the Bear Creek coal 38 in. deep, with an average air pressure of 2.6 in. under the grate, 118 tons of calcine were smelted per 24 hr., with 65 tons of coal (ratio 1.8); but with a fuel bed only 24 in. deep, and a 1.7 in. under-grate pressure, the decidedly better result was obtained of smelting 180 tons of calcine with 80 tons of coal (ratio 2.2). (See trials 6 and 7 in Table I.) A parallel run with 61 tons of Diamondville coal smelted 267 tons of calcine and flue dust (ratio 4.4), a normal result with this coal. (See trial 8.) The fuel cost for even the better of the Bear Creek results was 75 per cent. higher, per ton smelted, than for the Diamondville result, and would be nearly 100 per cent. higher if flue dust had been included in the charge, as it was with Diamondville, because of the slower smelting of dust.

Even with forced draft a really good flame could not be obtained from this lot of Bear Creek coal; but after 15 months further development of the mines the coal was of a quality to give fairly good flame without forced draft. The efficiency, however, was 25 per cent. better with forced than with natural draft, as indicated by the ratio and "efficiency" columns for trials 9 and 10 in Table I. Trial 10 was the most efficient and economical of all the tests of Bear Creek coal, but its fuel cost per ton smelted was 14 per cent. higher than for Diamondville in a parallel test (trial 11), and 30 per cent. higher than for normal Diamondville results when the charge, as in trial 10, contains no flue dust. Another

large shipment, six months later, was not as good as the one just mentioned (see trials 12, 14, 16, and 18 of the table.)

Another Montana mine-run coal, from Roundup, gave results which, as a whole, were about the same as with the Bear Creek (see trials 20 to 26), but the Roundup netted 5 to 10 per cent. less cost per ton smelted, because of a lower freight rate. The cost, however, averaged nearly 70 per cent. more with Roundup than with Diamondville coal, although the price of a ton of Roundup was one-sixth less than for Diamondville.

What Constitutes Good Fuel

So far as the coal is concerned, there are four factors that affect smelting efficiency: 1, the character of the true coal substance; 2, the percentage of fines; 3, the percentage of moisture; and 4, the character and percentage of ash; and these call for some discussion.

Character of the True Coal

As to the character of the real coal substance, something can be judged from proximate analyses; but these must be used with a good deal of caution, for they may be very misleading in regard to the ratio of fixed carbon to the volatile portion of the real coal, and they tell nothing as to the composition and qualities of the volatile matter. Frank Haas has shown that the two coals whose analyses are quoted below are undoubtedly identical in the character of the coal substance, though the proximate analyses suggest quite otherwise when considered in the usual way. They are from different parts of the same bed. His conclusion is based on intimate practical familiarity with the coals; on a study of the ultimate analyses; on consideration of the sulphur included as part of the volatile matter; and on changes in the ash which affect the apparent percentages of volatile matter and fixed carbon.[3]

Proximate Analyses

	Moisture	Volatile Matter	Fixed Carbon	Ash	Sulphur	Ratio of $\frac{\text{F. C.}}{\text{Vol.}}$
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
No. 2....	1.76	35.06	56.54	6.64	1.63	1.61
No. 9....	1.21	39.78	51.90	7.11	3.54	1.30

Ultimate Composition of the True Coal

	Carbon Per Cent.	Hydrogen Per Cent.	Nitrogen Per Cent.	Oxygen Per Cent.	Ratio of $\frac{\text{C}}{\text{H}}$
No. 2.....	82.76	5.82	1.69	9.73	14.2
No. 9.....	82.98	5.98	1.50	9.54	13.9

[3] *West Virginia Geological Survey*, vol. ii (A), p. 651 (1908); and *Bulletin* No. 10, Consolidation Coal Co., Fairmont, W. Va.

Mr. Haas's proximate analyses appear to have been made with a good deal of care; but with less care the results are still more misleading. The analyses in Table II for our trials 9, 10, and 11 illustrate what happens when the conditions, as to time and temperature, are not according to the standard directions for determining volatile matter. The ratio of volatile matter to fixed carbon, in these three cases, is respectively 1.01, 1.06, and 1.01; but in no other case in the table is the ratio higher than 0.89 for either of these coals. These three analyses were made together; and the fact that the ratio is equally high for both Bear Creek and Diamondville is practically conclusive evidence that the samples were heated too much in the determination of volatile matter. The errors were not appreciated till it was too late to make checks.

Even the moisture determination may lead to errors in the apparent percentage of volatile matter; P. L. Teed shows that in the standard method for moisture the loss from some coals amounts to 2 per cent. more than the true moisture.[4]

We know that while the volatile matter contains hydrocarbons, H and CO (combustibles), it also contains N, CO_2 and H_2O (non-combustibles), this H_2O having evidently been combined as part of the coal; and that the relative quantities of these different substances differ greatly from coal to coal. Porter and Ovitz found that the non-combustible gases from several coals (not including mechanically absorbed moisture), all treated in a similar way, varied from 1 per cent. to at least 15 per cent. of the total coal.[5] The ultimate analyses made by the government bureaus show oxygen content of unweathered bituminous coals, after reducing to an ash and moisture free basis, ranging from 5 to 17 per cent.; and in samples taken near the outcrop, where weathering has occurred, this figure runs as high as 27 per cent.[6] This oxygen not only displaces, so to speak, an equal amount of carbon and hydrogen, but also renders useless, for heat production, a considerable part of the carbon and hydrogen that are present, by being already combined with them. David White has shown, by examination of a very large amount of data, that each per cent. of oxygen has the same effect as 1 per cent. of ash in reducing the heating value shown by a calorimeter.[7] The commercial value is reduced in still greater measure, because the gas resulting from this oxygen has to be heated to the furnace temperature at the expense of the heat-producing portion of the coal, thus still further reducing the available heat and the margin above the critical temperature.

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[4] *Bulletin No.* 104, *Institution of Mining and Metallurgy* (May 15, 1913).

[5] *Bulletin No.* 1, *U. S. Bureau of Mines* (1910).

[6] *Bulletin No.* 22, *U. S. Bureau of Mines* (1913).

[7] *Bulletin No.* 382, *U. S. Geological Survey* (1909), reprinted as *Bulletin No.* 29, *U. S. Bureau of Mines.*

TABLE II.—*Reverberatory Coals Used at Anaconda*

No.	Kind of Coal.	Quality of Coal				
		Moisture	Volatile	Fixed Carbon	Ash	B.t.u. per pound
1.	Belt run of mine.....	8.5	23.6	38.7	29.2	8,180
2.	Diamondville run of mine..	8.0	36.6	45.4	10.0	11,180
3.	Belt run of mine	8.5	23.6	38.7	29.2	8,180
4.	Diamondville run of mine..	8.0	36.6	45.4	10.0	11,180
5.	Belt run of mine.....	6.7	23.7	38.7	30.9	8,340
6.	Bear Creek run of mine...	11.7	33.5	38.4	16.4	10,140
7.	Bear Creek run of mine....	11.7	33.5	38.4	16.4	10,140
8.	Diamondville run of mine..	6.7	39.7	45.3	8.3	12,310
9.	Bear Creek run of mine....	13.8	37.7	37.2	11.3	10,110
10.	Bear Creek run of mine....	13.1	39.4	37.1	10.4	10,340
11.	Diamondville run of mine..	6.9	41.4	41.0	10.7	11,470
12.	Bear Creek run of mine....	11.2	36.1	42.9	9.8	10,300
13.	Diamondville run of mine..	6.4	38.9	44.4	10.3	11,310
14.	Bear Creek run of mine....	11.2	35.6	43.7	9.5	10,300
15.	Diamondville run of mine..	6.5	39.4	44.9	9.2	11,600
16.	Bear Creek run of mine....	8.9	35.8	44.6	10.7	10,620
17.	Diamondville run of mine..	5.2	38.8	46.0	10.0	11,780
18.	Bear Creek run of mine....	9.7	37.1	43.3	9.9	10,700
19.	Diamondville run of mine..	6.4	39.5	44.2	9.9	11,550
20.	Roundup run of mine.....	13.6	30.5	46.2	9.7	10,320
21.	Diamondville run of mine..	6.4	37.5	44.7	11.4	11,440
22.	Roundup run of mine.....	13.3	31.2	45.3	10.2	10,420
23.	Diamondville run of mine..	5.8	36.9	47.6	9.7	11,780
24.	Roundup run of mine.....	12.5	30.6	45.9	11.0	10,620
25.	Diamondville run of mine..	5.3	38.5	45.7	10.5	11,460
26.	Roundup run of mine.....	13.0	31.8	45.1	10.1	10,300
27.	Diamondville run of mine..	9.1	36.2	45.0	9.7	11,820
28.	Bear Creek lump and Diamondville mine run.....	B.C. 9.7 Dia. 6.7	35.1 37.1	45.8 44.1	9.4 12.1	11,010 11,390
		Total....7.6	36.5	44.6	11.3	11,270
29.	Diamondville.....	6.7	37.1	44.1	12.1	11,390
30.	Diamondville lump.....	6.4	39.2	46.4	8.0	12,310
31.	Roundup lump.....	A.....17.3	31.6	43.1	8.0	10,500
		B.....19.7	31.2	41.8	7.3	10,320
		C.....18.4	30.8	43.3	7.5	10,440

In most bituminous coal there is about 5 per cent. of hydrogen (on the moisture-free basis), which of course burns to water, and this water must be considered when interpreting reports on the heating value. Laboratory determinations of heating value are made with this water condensed to liquid; but in actual use it passes away as gas, and the error in the reported heat units amounts to about 500 B.t.u. per pound of dry coal.

Manufacturers of illuminating gas find that the relative quantities of the different hydrocarbon gases and of the different tar-like substances volatilized from coal vary greatly at different temperatures with any particular coal, as well as from one coal to another: moderate temperatures yielding large quantities of the heavier products, while very high temperatures yield more of the lighter products. The lighter gases burn with a clear blue flame, while the heavier products burn with a more or less opaque yellow or white flame due to the incandescence of solid particles. All experience shows that this latter kind of flame is one of the chief essentials for successful reverberatory work, a very large part of the heat utilized by the charge being received by direct radiation from these highly incandescent particles. The trouble with a clear flame is that comparatively little of the hot gas comes in contact with the charge, to permit the direct absorption of heat; and not much of it comes in contact with the roof and walls, and so they receive comparatively little heat to radiate on to the charge; but the suspended incandescent particles have an immense surface per pound, from which they radiate a good deal of their heat to the charge. They also undoubtedly absorb some of the heat developed from the lighter gases and then radiate that to the charge also. With a good flame, the total surface of the incandescent particles in the furnace at any moment is far in excess of the entire surface of the roof and walls. The effect of this is like increasing the flow from a water reservoir, when a fire is to be extinguished, by multiplying the number of streams, and thereby greatly increasing the efficiency of action. This statement supplements the argument in regard to the "margin of temperature" on pp. 735 and 736. Two cases have already been cited in which a lack of incandescent flame was largely responsible for complete failure to do any smelting (Belt and Bear Creek coals). Another striking experience with this principle was had with the Brückner roasters at the old works, in which Belt coal gave very little flame at the fire-box end, and none at all at the other end even with forced draft. In order to get results it was necessary to drive the fire hard, which produced such a high temperature at the fire-box end that there was more or less trouble from sintering, while the charge never got hot enough at the other end to roast satisfactorily. With Diamondville coal this trouble did not exist. It should be noted that an *excessively* dense (smoky) flame is to be avoided, for two reasons: first, there is increased difficulty in supplying and mixing enough air for combustion; and second, it is almost certain that, in such a dense cloud, the heat radiated from most of the incandescent particles is obstructed and the charge not efficiently heated.

In order to get this luminous flame, even with good coal, a fairly thick fuel bed must be maintained on the grate; so that the condition is approximated in which just enough air comes through the grate to burn the fixed carbon to CO, while the rest of the air is admitted through adjustable

checker holes above the bridge wall. The large hearth forms nearly the ideal combustion chamber demanded by these conditions. In boiler practice it is rather unusual to have more than an 8-in. fuel bed; but in large reverberatories with 1½-in. draft in the flue a 24-in. bed is more or less standard for good coal containing only a moderate percentage of fines. With coals that do not flame well under natural draft there is often a marked improvement by using forced draft. The latter so increases the temperature in the fire box as to drive off more volatile matter. This is illustrated by the experiences with Belt, Bear Creek, and Roundup coals, all of which did better smelting with forced than with natural draft.

Percentage of Fines

In ordinary boiler practice, the equipment can be so designed and used as to get reasonably good efficiency with any coal that is fit to be so called, although good coal is much more satisfactory than poor, and is likely to be more economical; but in reverberatory smelting there are much narrower limitations. The chemical properties have been indicated as the most important of these; but experience proves that the range of size is also very important. On account of conditions that the Anaconda Co. could not control, the quality of coal received from Diamondville became worse several years ago, the chief difficulty being that it was necessary to mine a soft coal, which crumbles badly and yields a large percentage of fines. Our standard furnace results, with only a moderate proportion of fines, consist in burning 60 tons of coal per 24 hr. and smelting 270 tons of charge (ratio 4.5); 80 to 85 per cent. of the charge being hot calcine, and the rest flue dust, which is harder to smelt than calcine. The poor coal dropped the results to 240 tons smelted with 60 tons of coal (ratio 4.0). The coal at this time contained only about 40 per cent. coarser than ¾ in. as against 60 to 75 per cent. coarser than this size when getting normal results. A test of Diamondville lump, which probably contained 80 per cent. or more coarser than ¾ in., smelted 328 tons of calcine and flue dust with 63 tons of coal (ratio 5.2). (Trial 30 in Table I). Comparing ratios, this lump coal yielded 30 per cent. better results than the poor mine run, and 15 per cent. better than good mine run. Another test, with one-third Bear Creek lump and two-thirds of the poor Diamondville mine run, smelted 248 tons with 62 tons of the mixed coal (ratio 4.0); but a parallel test with the Diamondville alone smelted 254 tons with 59 tons of coal (ratio 4.3). (See trials 28 and 29.) The best result in three days' test of Roundup lump was 263 tons of calcine (no flue dust) smelted with 79 tons of coal (ratio 3.3). (Trial 31 C.) Comparing the ratio and "efficiency" columns of trials 20 and 31 C, the Roundup lump gave 40 per cent. better results than Roundup run of mine under similar conditions.

The three sets of results under trial 31 in the table show the need for the furnace men to learn by experience how best to handle each coal: there was a marked improvement each succeeding day, but we considered that the third day's result was the best that could be had with this coal. The shipment was not large enough for more than the three days' run; but in the other trials here recorded the preliminary experimenting was done before official testing was undertaken.

Neither the Bear Creek lump nor the Roundup lump is as economical as Diamondville run of mine. The Diamondville lump, however, is more economical; but, unfortunately, it cannot be supplied in sufficient quantity.

It may be stated that no coal larger than 7 or 8 in. ever goes into the fire box.

A coal sized anything like as closely as the regular commercial sizes of anthracite would probably give very poor results, because there would be such a free passage of air through the mass that the volatile matter would largely burn before it reached the hearth.

The reduced efficiency and increased cost resulting from a large percentage of fine coal are due to three considerations. First, the fresh supply of fine coal temporarily smothers the fire, thereby lessening the amount of air that can be drawn through; while the greater surface exposed per pound of fine than per pound of lump coal permits a sudden evolution of more volatile matter than can be burned by the available air supply, so that considerable fuel passes into the flue unconsumed; and to adjust the air supply through the checker work above the bridge is quite impracticable. Second, it is next to impossible to prevent a large loss of the fine coal through the grate. This loss at Anaconda is 25 to 30 per cent. of the coal when there is the excessive percentage of fines or an insufficient clinker bed, but only 12 or 15 per cent. under normally favorable conditions. Third, this dropping of fines is likely to leave large holes in the fuel bed for some time; but, as this condition does not occur till later than the excessive distillation of volatile matter, a good deal of unnecessary air enters and lowers the furnace temperature.

We are satisfied that the best solution of this question of fines is to pulverize all the coal and burn it by blowing it directly into the hearth of the furnace. Used in this way, all of the fixed carbon, and even the ash, as well as the heavy volatile hydrocarbons, supply incandescent surfaces for radiation of heat to the charge; there is absolutely no ash-pit loss; and the conditions are so uniform that the air supply can be adjusted to a nicety, with the assurance that there will be no decided shortage of air at one time and a considerable excess soon after. Moreover, there will be used only the minimum excess of air, so that a much greater "margin of temperature" will be attained. The extremely successful

results of this method at the Canadian Copper Co.'s plant, together with the results at several steel plants, are so encouraging that the Anaconda Co. is preparing to give it a thorough trial. The conditions necessary to success are much better understood than they were a few years ago, and we expect that there will be no commercially insurmountable difficulties.

Moisture

This is an impurity, which nobody wishes to pay for, and which requires heat to evaporate. Six per cent. of moisture in a pound of coal absorbs 120 B.t.u. in evaporating and being raised to the temperature of the reverberatory hearth; and in the case of Diamondville coal, burned in our large furnaces, reduces the temperature to 2,800° F. when it would otherwise be 2,825° F.; a reduction of the margin above the critical temperature that probably reduces the capacity and economy by *at least* 3 *per cent.*

Ash and Clinker

Ash, like moisture, reduces the heat units per pound of coal, and yet it has a distinct value: with a great many coals the loss through the grate would be excessive if there was not a bed of clinker. In some cases, however, the ash clinkers so badly as to seriously clog the grate and interfere with combustion. This was the case with Diamondville coal when it was used under forced draft in 50-ft. furnaces at Anaconda. The average treatment of calcine, flue dust, and added lime rock, with forced draft, was 104 tons per 24 hr., while using 44 tons of coal (ratio 2.4); but with natural draft it was 144 tons of similar charge, using 46 tons of coal (ratio 3.1); an increase of 30 per cent. in efficiency. A little of this increase was due to certain changes in furnace design; but, as intimated, most of it resulted from the adoption of natural draft. The most important change in design was, in fact, a 58 per cent. increase in the grate area (from 53.3 sq. ft. to 84 sq. ft.) to suit the requirements of the coal under natural draft. With forced draft there was a tremendous volume of dense flame for some time after the grate was cleaned; so much, in fact, that a good deal of fuel passed away unburned. This excessive flaming was due to the high temperature in the fuel bed, caused by the intense forced-draft combustion; but this high temperature gradually covered the grate with a dense clinker, so that the passage of air was much reduced and there was practically no flame in the furnace. The blast had to be shut off every 4 hr. to clean the grate. An attempt to improve matters by introducing steam with the blast failed to help much, though with Belt coal steam was of marked benefit in reducing clinker troubles. A large percentage of ash prevents efficient contact of air with the fuel in the fire box, unless there is a very deep bed. When using Belt coal (25 to 30

per cent. ash) with natural draft and only a 24-in. bed, it is altogether probable that there was a large excess of oxygen in the hearth (though we have no gas analyses to confirm it), for a good deal of the air passing through the bed would come in contact with very little except ash. There was not enough volatile combustible to utilize this extra air (the ratio of volatile matter to fixed carbon is low in this coal), which therefore acted to cool the hearth. With the deep-bed gas-producer conditions this excess air could not pass through. We have gas analyses to prove this.

The essence of the clinkering problem is, like that of the slag problem, the question of fusibility; but it is seldom possible to predict from analysis of the ash what will be the qualities of the clinker, or even whether it will or will not clinker. For example, there are decided variations in the four following analyses of Diamondville ash, yet the clinkering qualities are practically constant.

SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	Alkalies	SO_3
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
37.5	6.8	23.6	16.6	2.0	6.3	6.0
50.3	7.4	25.4	9.4			
51.9	7.0	21.6	12.3			
60.0	9.6	19.6	7.4			

It often happens that there are great variations in clinkering properties of different portions of the ash; yet if one portion is easily fusible, at the temperature to which it is subjected after the fuel is largely burned away from it, it is quite certain to surround other portions, so that they contribute to the volume of clinker though they have not fused. The most satisfactory investigations of the clinker problem seem to be those made by E. G. Bailey.[8] He shows the present hopelessness of judging anything from analyses; but has adopted a laboratory fusion test of the average ash, and says:

"Even if the ash is made up of a mixture of high and low fusing temperature material and if only that having a fusing temperature below that of the fuel bed is melted into a clinker, there will be a greater percentage of such ash fused into a clinker from a coal having a low average fusing temperature than from a coal having a high average fusing temperature. In actual practical tests it has been found that the percentage of ash which is formed into clinker as well as the obstructed grate area per pound of clinker hold a close relation to the fusing temperature of the ash from different coals when burned under similar conditions."

He shows plots of some results which indicate that, with good average conditions on steam-boiler grates, less than 25 per cent. of the total ash clinkers if its average fusion temperature is 2,500° F. or higher; but the proportion of clinker rapidly increases to 50 per cent. as the average fusion temperature falls to 2,300° F.

[8] Ohio Society of Mechanical, Electrical and Steam Engineers, Nov. 18, 1911; and *Bulletin No. 5*, Fuel Testing Co., Boston.

Description and Discussion of Coals Used

The Diamondville run of mine is decidedly the most economical of all the coals we have used, though it costs, delivered to the furnaces, from one-eighth to one-half more than any of the others except Rock Springs. This is because it is much the most efficient. It is a long-flame bituminous coal from which, in a properly designed furnace, with natural draft, the volatile combustible matter is distilled in just about the same length of time that the fixed carbon is burning in the fire box; keeping the entire furnace nicely filled with actively burning flame about half the time, and a very considerable amount of flame all the time. The distillation is rapid enough to produce a high temperature near the bridge, where the ore is charged, yet slow enough for the gases to give up a reasonable proportion of their heat before passing into the flue. The normal furnace results consist in smelting 270 tons of hot calcine and flue dust per 24 hr., while using 60 tons of this coal. The temperature of the gases averages 2,800° F. just beyond the bridge, while the formation temperature of the slag is about 2,000° F. The freezing temperature of the slag, after it is formed, is possibly not over 1,800° F., but it has an actual temperature of nearly 2,100° F. as it runs out of the flue end of the furnace. The gases enter the flue at an average of 2,200° F. These conditions guarantee good settlement of the matte, and easy skimming of the slag.

With good natural draft the Diamondville ash forms a clinker of moderate hardness, which is barred out without much trouble, and at the same time maintains an excellent fuel bed and keeps the total loss of fuel, through the grate, down to 12 or 15 per cent. when there is not an excessive amount of fines in the coal received. Most of this loss occurs at the time of grating. With forced draft, however, the clinker is so hard and tough, and forms in such large masses, as to require excessive labor in cleaning the grates. It is thus seen that, with natural draft, Diamondville approaches the ideal coal for smelting; and in fact it has the reputation of being one of the best coals available in the Rocky Mountain region for this purpose.

Belt is a bituminous coal, containing a good deal of carbonaceous shale and considerable iron pyrite, the 25 and 30 per cent. of ash in the mine run being a serious source of difficulty and inefficiency in its use. With moderate fire-box temperatures the ash does not clinker, but remains to a large extent in lumps; yet with the intense fire-box combustion accompanying forced draft, and sometimes with strong natural draft, it clinkers seriously unless steam is used beneath the grate. It ordinarily gives but very little good flame unless the temperature in the fuel bed is greatly increased, by means of forced draft, and even then the flame is far from ideal. Not only is the percentage of volatile matter low, but its ratio to fixed carbon is also low. When used in our regular fire box with

natural draft it will do no smelting; and even with the immense gas-producing fire box the temperature a little beyond the bridge was only 2,500° to 2,600° as compared to 2,800° from Diamondville coal used in the regular fire box; but there were very large volumes of this lower-temperature gas, because of the great quantity of coal consumed.

Bear Creek is a sub-bituminous coal (between good bituminous and lignite). It flames sufficiently to smelt with natural draft on a normal-sized grate; but gives a better flame and more efficient smelting with forced draft. Under any conditions it flames, after fresh firing, only a quarter to a half as long a time as Diamondville, and so requires more frequent firing. With natural draft it produces a temperature just beyond the bridge 150° to 200° F. lower than Diamondville; and with forced draft it is 100° F. lower than Diamondville. It appears to deteriorate in heating value rather fast after mining, but does not slack seriously. It does not clinker much under either natural or forced draft; and, for that reason, is subject to a 25 per cent. fuel loss through the grate.

Roundup is also a sub-bituminous coal. It flames similarly to Bear Creek and gives about the same furnace temperatures. It slacks somewhat rapidly on exposure. Its ash clinkers somewhat more than Bear Creek, but not as much as Diamondville; and consequently suffers an intermediate fuel loss into the ash pit.

It seems reasonable to suppose that, after removing moisture, the higher the ratio of hydrogen to carbon the larger will be the proportion of total combustible that will be given off in the volatile form and serve to produce flame in the hearth; other conditions, of course, being the same. On the other hand, flaming is in all probability reduced the higher the ratio of oxygen to total combustible. If these two statements are true, then the relation between the first of these ratios and the second is one indicator as to the value of coals for smelting. The following figures are derived from average ultimate analyses of certain of the coals discussed in this paper. These analyses are taken from *Bulletin No.* 22 of the U. S. Bureau of Mines (samples from near the surface of mine workings being omitted); and the relation just mentioned is shown in the last column.[9]

	1	2	3	4	5	Ratio of Column 4 to Column 5
	H	C	O	$\frac{H}{C}$	$\frac{O}{H+C}$	
	Per Cent.	Per Cent.	Per Cent.			
Diamondville	5.33	76.89	11.16	0.069	0.14	0.49
Rock Springs	4.96	74.28	13.72	0.067	0.17	0.39
Bear Creek	4.94	66.78	14.04	0.074	0.20	0.37
Belt	3.64	61.23	11.50	0.059	0.18	0.33

For these four coals the numbers in the last column stand in the same order as the normal smelting efficiencies that can be obtained with any

[9] The *Bulletin* gives no ultimate analyses of Roundup coal.

ordinary provision short of the elaborate and expensive gas-producing fire box that was used for Belt.

Other Important Factors

Mixing the Charge

Of the factors other than fuel, the one that requires most careful attention is proper fluxing of the charges: and this means, not merely that a day's average sample shall have a suitable analysis, but that each individual charge shall have such composition as to yield a readily fusible and fluid slag, of low specific gravity; it also means that the speed of smelting depends on how intimately the various constituents of each charge are mixed. From the following facts it is clear that these conditions are not always attained. In December, 1903, Mr. Mathewson changed the practice of adding a small proportion of lime rock flux directly to the reverberatories, without any special mixing with the rest of the charge, to mixing it, with some regularity, with the concentrates and fine ore fed to the roasters. The result was to increase the reverberatory performance from 144 tons smelted with 46 tons of coal (ratio 3.1) to 163 tons smelted with 48 tons (ratio 3.4). The most striking example we have had of the effect of poor fluxing came when a charge of calcine failed to smelt in three times the usual 1 hr. and 20 min., though the entire contents of the hearth got so hot that the matte pool "boiled" (gave off sulphur) at the end of the 4 hr., and frothed considerable of the accumulated slag out of the furnace. It is, of course, understood that any approach to such poor results is not at all common.

Preheating the Charge

It is well known that the capacity and economy of a furnace are very much decreased if the hot calcine from the roasters is allowed to cool before smelting. At Argo, Colo., a 23 per cent. increase in capacity was obtained, with no increase of fuel consumption, by charging the hot calcine directly to the reverberatory instead of allowing an intermediate cooling. The calcine made up only one-half of the charge, the other half continuing to be cold ore.[10] At Anaconda the calcine enters the reverberatories at a temperature of 900° F.; and most of the flue dust is thoroughly hot.

Preheating the Air

Various attempts to save money by preheating the air for copper blast furnaces have been abandoned, though the fuel efficiency of the furnaces themselves was much increased when the air was raised to 700°

[10] Peters's *Modern Copper Smelting*, p. 446.

or 800° F. There have been two reasons for the commercial failure: first, the cost of the extraneous fuel used to do the heating; second, the cost for much extra power to force the blast through the small and tortuous passages of the heating devices. Both of these troubles can be overcome for reverberatory work. The first step is to use the waste heat from the furnaces, instead of extra fuel. The gas comes away from these furnaces at a much higher temperature than from blast furnaces, and there is not the frequent inrush of large volumes of cold air such as occurs through the charge doors of the blast furnaces. In the few cases where preheated air has been used in reverberatories for copper smelting, natural draft and efficient heating devices have been utilized. The gas-fired furnaces at Great Falls have always had the checker work of the open-hearth steel furnace. The Peyton Chemical Works, near Martinez, Cal., heated the air for an oil-fired furnace in open flues, without checker work but with several turns purposely introduced. With hot air (varying from 750° to 1,830° F.) their furnace capacity was 50 per cent. greater than with cold air. It is easier and cheaper to remove dust from these open flues than from the checker work.[11]

Producing Steam with Waste Heat

Since the practice of utilizing the flue gases to generate steam was adopted at Anaconda it has also been put into effect at several other large plants. At Anaconda about 25 per cent. of the heating value of the total coal is thus made effective; while only 15 per cent. of this value is actually utilized for smelting. This steam from the eight furnaces generates 30 per cent. of the 16,700 mechanical horse power used in the entire concentrating and smelting plant.

Boilers for this service must be arranged for the gases to pass through in a straight line, and with plenty of passage area, in order not to affect the draft. With the several changes of direction found in ordinary boiler practice the smelting capacity is seriously reduced; so much, in fact, as to affect the profits much more than the money value of all the steam.

For a good deal of information and suggestion, the writer is indebted to W. M. Kelly, general smelter foreman; F. W. C. Whyte, manager of coal mines; and especially to Edward O'Brien, general reverberatory foreman.

[11] Fred A. Leas: *Engineering and Mining Journal*, vol. lxxxvi, No. 19, p. 898 (Nov. 7, 1908).

The Annealing of Cold-Rolled Copper

BY EARL S. BARDWELL, GREAT FALLS, MONT.

(Salt Lake Meeting, August, 1914)

THE determination of suitable and safe annealing temperatures is one of the most important problems arising in the operation of a copper rolling mill. Certain of the larger mills have worked this problem out for themselves and in these mills the operation has become more or less standardized. In some of the mills pyrometers are used to measure the temperature of the annealing furnaces and careful watch is kept to see that overheating does not take place. In a considerable number of the mills, however, dependence is placed entirely on the judgment of the operator. The almost total lack of literature bearing on this subject seems sufficient warrant for the publication of this paper.

The discussion of this subject naturally divides itself into two parts: (1) The effect of annealing temperature on physical properties; and (2) changes in the microstructure of refined copper due to cold rolling and annealing.

I. THE EFFECT OF ANNEALING TEMPERATURE ON PHYSICAL PROPERTIES

For the purpose of investigating this phase of the subject six test bars were selected, varying in oxygen content between the limits of 0.036 and 0.070 per cent. The chemical composition of each of these test bars was as follows:

	Bar Number					
	1	2	3	4	5	6
Copper + silver	99.95	99.92	99.945	99.94	99.95	99.93
Oxygen	0.041	0.070	0.046	0.050	0.036	0.058
As + Sb	0.0038	0.0038	0.0038	0.0038	0.0031	0.0031

The bars were heated in an electric muffle to about 1,650° F. and rolled into rods. The rods were annealed, pickled, and drawn to No. 12 B. & S. gauge wire. The wire which was to be used for the annealing test was cut into 38-in. lengths and each length made into a coil 6 in. in diameter. The separate coils were marked so that they might readily be identified

as they were taken out of the annealing furnace. For annealing the wires an electric muffle was used and the annealing temperatures were measured by means of a thermo-couple. The temperatures as measured are accurate to about 10° and we were able, in practically every case, to so regulate the furnace that the temperature did not rise more than 20° above or fall more than 20° below the desired temperature while the wires were in process of being annealed. The muffle was brought to the desired temperature and six wires, one from each of the test bars, were introduced. The wires were allowed to remain in the furnace for 20 min., at the end of which time they were removed and plunged into a dilute sulphuric acid solution to remove scale. In this manner wires from each of the test bars were annealed at a series of temperatures varying between the limits of 300° and 1,800° F.

Electro-Conductivity of Annealed Wires.—The conductivities of the wires were measured by means of a Hoopes bridge manufactured by the Leeds-Northrup Co., of Philadelphia. The results of the conductivity tests are given in Table I.

TABLE I.—*Conductivities of Copper Wires Annealed at Various Temperatures for 20 Min.*

Temperature,	Per cent. Conductivities of Wires Drawn from Bar Numbers					
Degrees F.	1	2	3	4	5	6
60	97.3	96.2	97.6	96.9	97.2	97.0
300	97.45	96.35	97.6	97.1	97.4	96.9
400	97.5	96.4	97.8	97.1	97.5	97.1
525	97.8	96.6	97.9	97.3	97.7	97.4
625	97.9	97.2	98.0	97.5	97.9	97.4
700	99.5	99.0	100.0	100.0	99.9	99.8
800	100.3	99.6	100.4	100.0	100.3	100.1
900	100.2	99.6	100.4	100.0	100.3	100.1
1,000	100.3	99.5	100.3	100.0	100.2	100.1
1,100	100.3	99.3	100.3	99.9	100.1	99.9
1,200	100.2	99.3	100.3	100.0	100.1	99.9
1,300	100.1	99.2	100.3	99.9	100.2	99.8
1,400	100.1	99.1	100.3	100.0	99.9	99.5
1,500	99.85	98.7	100.1	99.6	99.65	99.2
1,600	99.4	98.5	99.65	99.1	99.25	98.7
1,700	98.9	97.6	98.7	98.3	98.3	98.1
1,800	97.8	96.7	98.0	97.3	97.0	97.0

The per cent. conductivity is referred to the annealed copper standard, which at 20° C. is represented by a resistivity of 0.15328 ohm (meter, gram). The conductivities reported at 60° F. are the conductivities of the hard-drawn wire. The low conductivities in the case of wires annealed at temperatures between 300° and 700° F. are due to imperfect or incomplete annealing, whereas the low conductivities obtained in the case of wires annealed at the higher temperatures are due to changes in

the physical structure of the copper brought about by annealing at these temperatures.

The curves, Fig. 1, are plotted with conductivities as ordinates and temperatures at which the wires were annealed as abscissæ. It will be noted that until an annealing temperature of about 600° F. is reached the conductivity increases very gradually; between 600° and 800° F. a very rapid increase in conductivity takes place. Wires annealed at 800° F. and at higher temperatures up to 1,200° F. show very little or no decrease in conductivity. Wires annealed at temperature higher than 1,200° F. in general show a decrease in conductivity, slight at first, but increasing rapidly until in the case of wires annealed at 1,800° F. the conductivity has become very nearly equal to that of the hard-drawn wire.

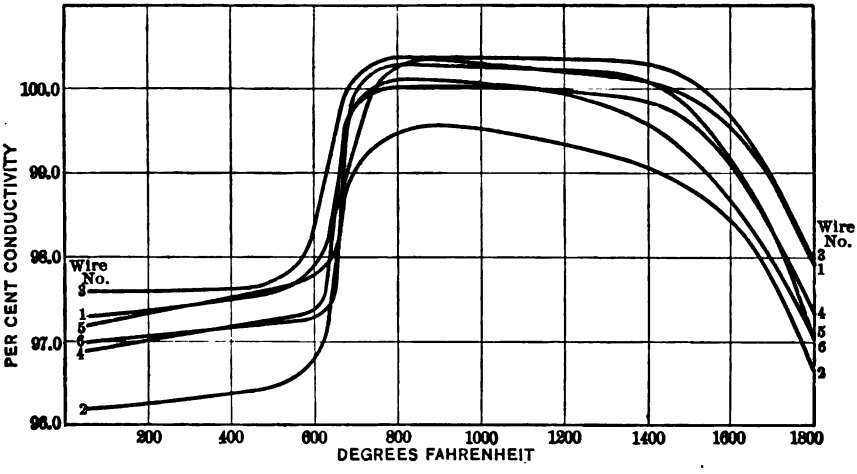

FIG. 1.—VARIATION IN CONDUCTIVITY OF COPPER WIRES WITH VARIATIONS IN ANNEALING TEMPERATURE.

It is interesting to note that wires drawn from bars 1, 3, 4, and 5 show no considerable decrease in conductivity until an annealing temperature in excess of 1,400° is attained, whereas the wires drawn from bars 2 and 6 show a marked decrease in conductivity when annealed at a temperature higher than 1,200° F. This is more especially marked in the case of the wire drawn from bar 2, which bar is the higher in oxygen.

Tensile Strength and Elongation of Annealed Wires.—After testing the annealed wires for conductivity they were tested for tensile strength and elongation. In making the tensile-strength determinations 1-ft. lengths of each of the various wires were used and the elongation measured to thousandths of an inch. Slight imperfections in the wire are responsible for the erratic results obtained in the case of several of the wires. Results which were clearly due to flaws in the wire have been disregarded.

Table II gives the results of the tensile-strength tests. In Fig. 2 these results are shown graphically.

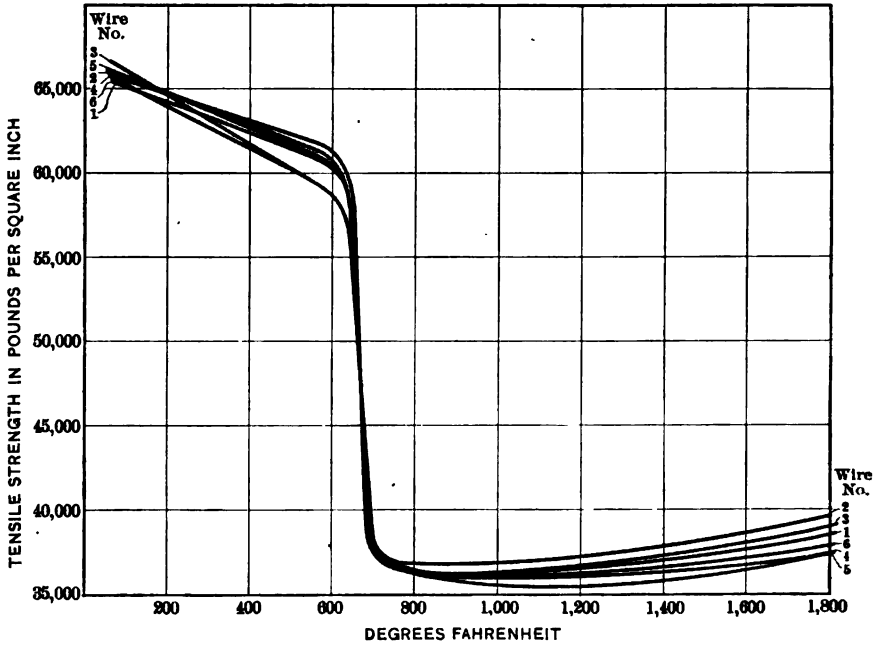

Fig. 2.—Variation in Tensile Strength of Copper Wire with Variations in Annealing Temperature.

Table II.—*Tensile Strength of Copper Wires Annealed at Various Temperatures for 20 Min.*

Temperature, Degrees F.	Tensile Strength, Pounds per Square Inch, of Wires Drawn from Bar Number					
	1	2	3	4	5	6
60	63,100	65,700	66,700	66,100	63,700	65,600
300	62,800	62,500	63,500	60,000	63,800	63,800
400	61,200	61,750	61,900	59,900	61,700	62,500
525	61,200	61,750	59,200	59,800	61,800	61,800
625	.....	60,750	58,100	57,400	62,500	.....
700	39,400	37,800	38,200	37,800	35,800	37,400
800	36,200	36,400	36,200	37,000	37,000	36,600
900	35,800	37,000	36,200	36,200	33,500	36,200
1,000	36,900	37,000	36,800	36,200	36,000	36,200
1,100	35,800	35,600	36,400	36,200	35,400	36,200
1,200	36,400	37,000	36,400	35,800	35,100	36,600
1,300	34,700	36,800	37,200	35,200	35,500	35,600
1,400	36,900	35,600	37,400	36,400	36,200	.....
1,500	36,400	.....	38,400	37,600	35,800	36,400
1,600	38,200	38,700	37,400	35,200	37,700	37,800
1,700	38,100	38,600	38,000	36,900	37,400	38,900
1,800	38,400	39,800	39,200	37,200	37,300	37,200

The tensile strength was figured by multiplying the breaking weight in pounds by the reciprocal of the cross-sectional area in square inches. The scale formed by annealing and subsequently removed by pickling caused a considerable reduction in the cross-sectional areas of wires annealed at the higher temperatures.

The curves show essentially the same characteristics.

In the range between 300° and 600° there is a slight decrease in tensile strength, followed by a sudden drop in the narrow range between 600° and 700°. Between 700° and 1,400° there is very little change in the tensile strength, which reaches a minimum at about 1,100°. Beyond 1,400° the tensile strength seems to increase slightly.

Table III gives in detail the per cent. elongation obtained in the case of each wire.

TABLE III.—*Per Cent. Elongation of Copper Wires Annealed at Various Temperatures for 20 Min.*

Temperature,	Per cent. Elongation of Wires Drawn from Bar Number					
Degrees F.	1	2	3	4	5	6
60	0.7	0.9	1.0	0.9	1.2	1.0
300	1.7	1.9	1.7	1.5	1.9	1.6
400	1.7	1.9	2.3	2.1	1.7	1.9
525	1.6	2.3	1.8	2.1	2.0	2.0
625	....	2.5	2.7	1.8	2.0	1.8
700	21.2	39.4	35.2	29.6	26.1	28.5
800	46.0	42.6	48.0	42.4	42.4	40.5
900	46.8	....	43.2	43.6	41.2	41.8
1,000	42.7	43.2	44.5	42.7	41.1	41.2
1,100	46.2	....	45.3	43.8	41.8	40.0
1,200	42.5	....	49.7	41.2	42.5	42.1
1,300	43.7	42.0	43.6	43.5	41.9	41.8
1,400	41.7	42.8	40.8	40.1	41.4	40.0
1,500	37.7	....	39.0	38.4	34.8	37.9
1,600	32.9	37.2	....	....	36.8	35.2
1,700	34.6	34.0	32.9	32.3	33.1	34.4
1,800	26.3	25.6	28.7	30.0	28.3	27.4

These results are shown graphically in Fig. 3. As will be noted upon examining the curves, there is a slight increase in the per cent. elongation as we pass from 300° to 600°, followed by a rapid increase in the range between 600° and 800°. From 800° to 1,300° the per cent. elongation remains fairly constant, reaching a maximum at about 1,100°, after which it decreases slowly at first and then more rapidly until at 1,800° the elongation is from 15 to 20 per cent. lower than in the case of wires annealed at 1,100°.

In Table IV the results of physical tests on the six wires annealed at different temperatures will be found averaged.

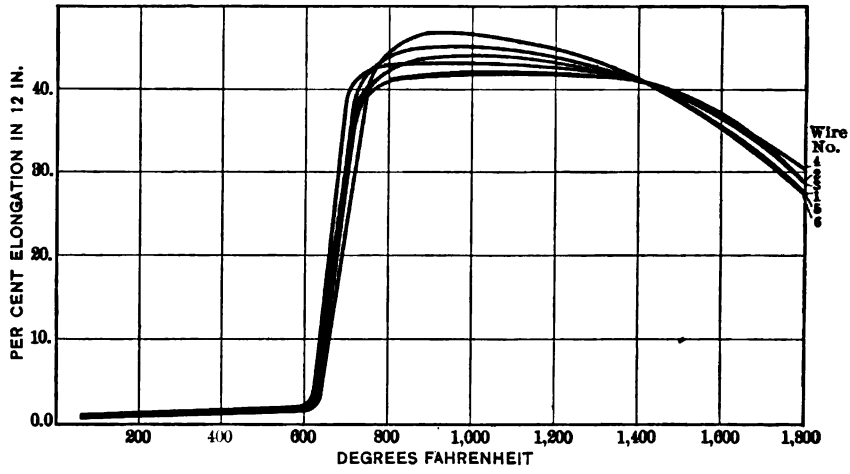

FIG. 3.—VARIATION IN PER CENT. ELONGATION OF COPPER WIRES WITH VARIATIONS IN ANNEALING TEMPERATURE.

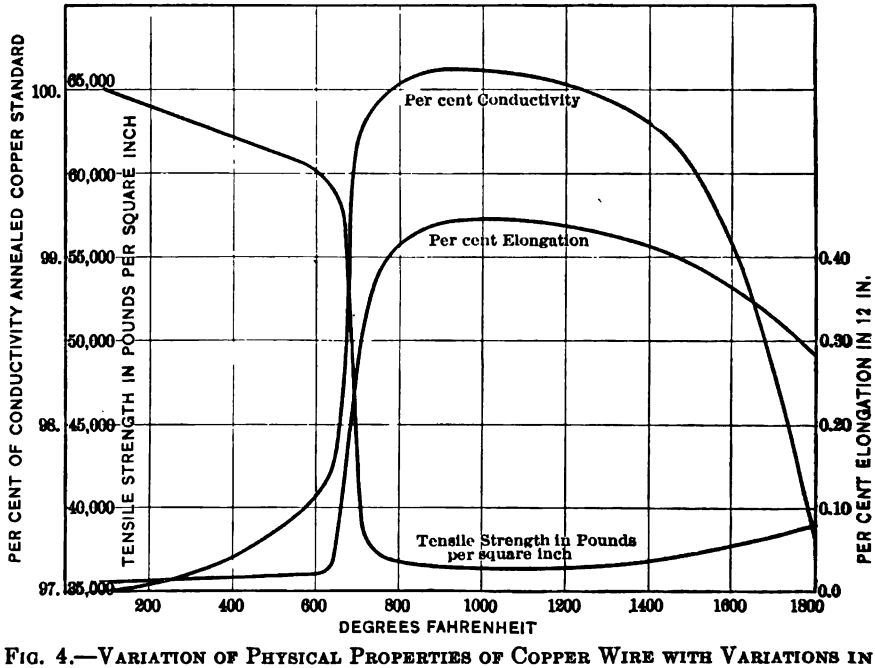

FIG. 4.—VARIATION OF PHYSICAL PROPERTIES OF COPPER WIRE WITH VARIATIONS IN ANNEALING TEMPERATURE.

TABLE IV.—*Composite Tabulation of Results of Physical Tests on Wires Annealed at Different Temperatures*

Temperature, Degrees F.	Conductivity, Per Cent.	Tensile Strength, Pounds per Square Inch	Elongation, Per Cent.
60	97.0	65,150	0.9
300	97.1	62,750	1.7
400	97.2	61,490	1.9
525	97.4	60,925	2.0
625	97.6	59,690	2.1
700	97.7	37,730	30.0
800	100.1	36,570	43.6
900	100.1	35,820	43.3
1,000	100.1	36,520	42.5
1,100	100.0	35,930	43.4
1,200	100.0	36,220	43.6
1,300	99.9	35,820	42.7
1,400	99.8	36,500	41.1
1,500	99.5	36,920	37.5
1,600	99.1	37,500	35.5
1,700	98.3	37,980	33.5
1,800	97.3	38,180	27.7

These data are plotted in Fig. 4. The curves show that the range between 800° and 1,300° F. is best suited for annealing. Wires annealed at around 1,100° F. have the maximum conductivity, maximum per cent. elongation, and minimum tensile strength.

Captain C. Grard, of Paris, France[1] has carried out a series of tests with reference to the effect of various annealing temperatures on cold-worked brasses of various compositions and copper. This investigator has not studied the effect of annealing temperature upon the electro-conductivity of the copper; he has, however, studied the effect of the annealing temperature upon the tensile strength and elongation of cold-worked copper. Fig. 5 is a reproduction of the curves given in Grard's paper. In Fig. 5 the temperatures are expressed in degrees centigrade, whereas in Figs. 1, 2, 3, and 4 the temperatures are given in degrees Fahrenheit. While the curve for elongation is of the same general shape as that obtained by us, the transition which we have found to take place between 600° and 800° F. he has found to take place between 150° and 200° C., or in the range between say 302° and 392° F. This may be partly due to the fact that the copper which Grard used for his experiments was much higher in oxygen than that used by us. As nearly as can be ascertained from Grard's paper, the copper used for his experi-

[1] Captain C. Grard, Paris: Industrial Application of Microscopic Metallography for Controlling the Work put on Copper and Brasses, International Association for Testing Materials, vol. ii, No. 11, II_{18}.

ments contained 0.15 per cent. oxygen. Grard does not state the length of time that each specimen was subjected to the annealing temperature. The difference noted between his curves and ours may be due to specimens being annealed for longer periods at the lower temperatures than was the case at higher temperatures.

Grard divides his curve into five zones. From 0 to 150° C. he calls the "zone of cold working," from 150° to approximately 200° C. is the "relaxed zone," from 200° to 400° the "zone of recuperation," from 400° to 650° is the "zone of complete annealing," and from 650 ° C. up is the "bending zone." Converting degrees centigrade into degrees Fahrenheit we have:

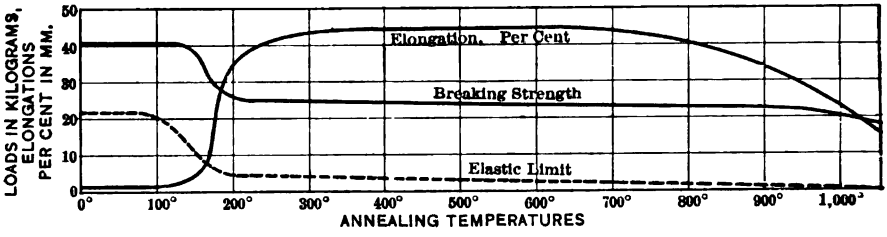

FIG. 5.—GRARD'S CURVES SHOWING EFFECT OF ANNEALING TEMPERATURE ON PHYSICAL PROPERTIES OF COPPER.

Zone of cold working.....	32° to 302° F.
Relaxed zone.....	302° to 392° F.
Zone of recuperation.....	392° to 752° F.
Zone of complete annealing.....	752° to 1,202° F.
Bending zone.....	1,202° to melting point of copper

Using the same terminology, the curves for per cent. elongation shown in Fig. 3 might be divided into zones as follows:

Zone of cold working.....	32° to 600° F.
Relaxed zone.....	600° to 750° F.
Zone of recuperation.....	750° to 900° F.
Zone of complete annealing.....	900° to 1,200° F.
Bending zone.....	1,200° to melting point of copper

This is, of course, only approximate. It is extremely difficult if not altogether out of the question to determine with any considerable degree of accuracy the per cent. elongation of a 1-ft. length of No. 12 copper wire that has not been annealed. The results which have been obtained are consistent and serve our purpose. As will be noted, the point which we have fixed upon as marking the beginning of the bending zone is the same point that Grard has fixed upon. The electroconductivity can be measured with a much greater degree of accuracy than can the per cent. elongation, and the conductivity of a copper wire is perhaps an even more delicate index of the changes that take place in

the copper upon annealing. If we base the limits of the several zones on the conductivity curves (see Fig. 1) we get results somewhat as follows:

Zone of cold working.....	32° to 300° F.
Relaxed zone.....	300° to 675° F.
Zone of recuperation.....	675° to 800° F.
Zone of complete annealing.....	800° to 1,100° F.
Bending zone.....	1,100° to melting point of copper

Grard states that the first two zones should never correspond to a finished state of the worked metal, as the metal in either of these zones is in a state of unstable equilibrium. He states further that the so-called bending zone should be avoided. This means that copper which attains in the annealing furnace a temperature higher than 1,100° to 1,200° F. has been overheated. Where copper is to be cold rolled after annealing a temperature of 1,100° F. will be found to give most satisfactory results. I do not wish to convey the idea that heating to higher temperature than 1,100° F. invariably injures the copper. Such is not the case. The nature of the troubles that may be brought about by overheating will be discussed on another page.

2. Changes in the Microstructure of Refined Copper due to Cold Rolling and Annealing

Fig. 6 is a photomicrograph of a piece of copper that has been cold rolled from $\frac{3}{16}$ to 0.033 in. In the photomicrograph note the slip bands, very fine parallel lines at right angles to the direction of rolling. These slip bands are characteristic of a strained condition of the metal resulting from severe mechanical treatment. In order to illustrate the affect of annealing upon the microstructure of cold-rolled copper, pieces of the above sheet were annealed in a small electric furnace at a number of different temperatures, each specimen being allowed to remain in the furnace for 20 min. The temperatures were measured by means of a thermo-couple standardized by the United States Bureau of Standards and are correct to 5° C. The temperatures, which were measured in centigrade degrees, have been converted to Fahrenheit degrees in order to avoid confusion and to facilitate comparison of microstructure with physical properties as shown by the curves in Figs. 1, 2, 3, and 4.

Fig. 7, a photomicrograph of a piece of the above-mentioned cold-rolled copper annealed for 20 min. at 392° F., shows the metal to be still in a strained condition. Under the microscope we see no evidence of crystals. The slip bands noted in Fig. 6 are visible also in Fig. 7.

In Fig. 8 we note a change. The slip bands have disappeared and in their place we have small crystals very closely grouped. We have now passed out of the zone of cold working and have entered the relaxed zone, which may be said to be characterized by very small crystals.

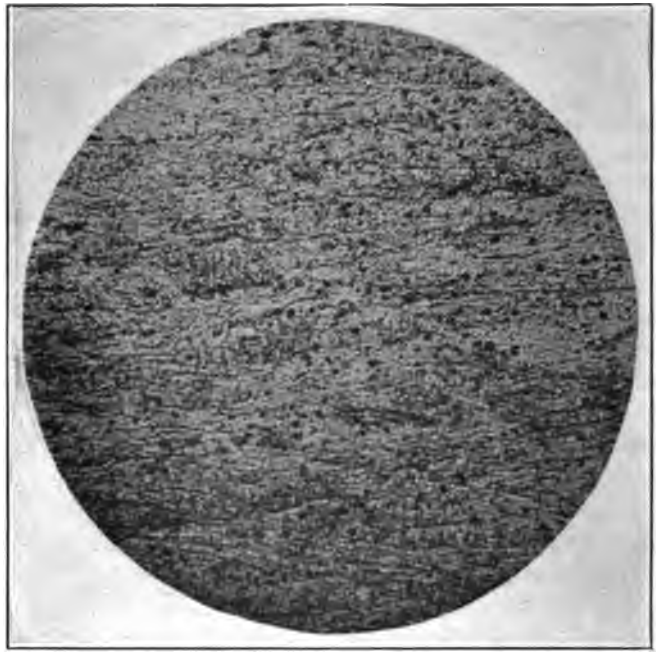

FIG. 6.—SHEET COPPER AFTER COLD ROLLING FROM 3/16 IN. TO 0.033 IN. × 100.

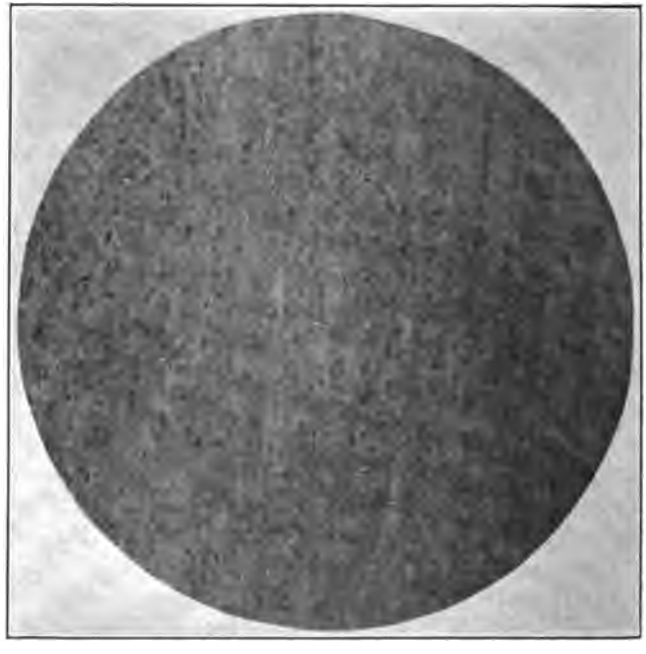

FIG. 7.—AFTER ANNEALING AT 392° F. × 100.

Fig. 9 is a photomicrograph of a specimen of the cold-rolled sheet which has been annealed for 20 min. at 752° F. This photomicrograph is characteristic of the zone of recuperation. The crystals are still very small, but are slightly larger than those shown in Fig. 8 and more regular in shape.

With Fig. 10 we enter the zone of complete annealing. The crystals in this specimen, which was annealed at 932° F., are much larger and more regular in outline. With Fig. 11, 1,112° F., they have reached their maximum regular growth. From this point on the crystals increase very

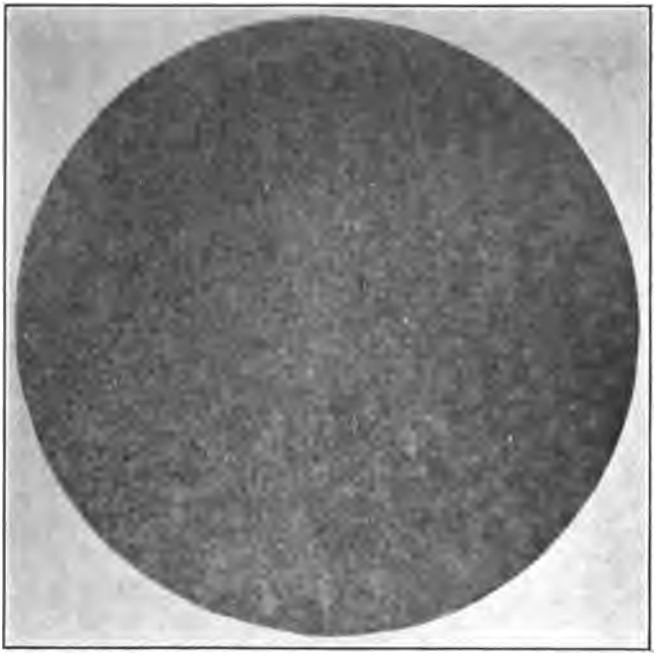

FIG. 8.—AFTER ANNEALING AT 572° F. × 100.

rapidly in size but they grow at the expense of one another and hence become irregular in size; *i.e.*, some crystals are very large while others are much smaller. This irregular growth, and the further fact that the crystal outlines are becoming more and more rounded in the case of specimens annealed at the higher temperatures, probably explain in part the decrease in ductility which we have noticed in the case of the wires annealed at different temperatures. In Figs. 12, 13, 14, and 15 the crystals become successively larger, the crystal faces become more and more curvilinear in outline, and polysynthetic twin crystals make their appearance.

The microstructure of refined copper that has been annealed after

rolling differs from that of the cast metal by the parallel markings (polysynthetic twin crystals) which appear on the crystal faces (see especially Figs. 14 and 15). The significance of these twin crystals is not entirely clear. The distortion of the metal brought about by rolling results in a refinement of the crystalline structure and an increase in tensile strength. This refinement of the crystal structure is brought about by sliding taking place along the cleavage planes of the polygonal grains. As a result the new crystals become arranged in twinning position with their neighbors. The temperature at which the strained metal is annealed seems to exert a considerable influence upon the formation of these poly-

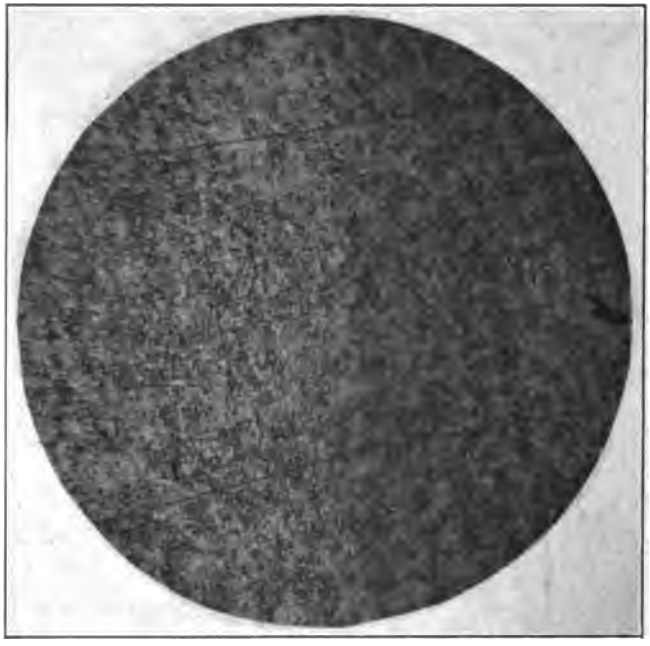

FIG. 9.—AFTER ANNEALING AT 752° F. × 100.

synthetic twin crystals as well as upon the size of the crystals in general. This increase in the size of the crystals is accompanied by a decrease in the per cent. elongation. The decrease in conductivity and in general the changes which take place in the case of the physical properties of wire annealed at the higher temperatures show a reversion of the metal to a condition very much like that of the cast metal. It is of interest to note that the appearance of the so-called polysynthetic twin crystals is coincident with the decrease in conductivity and per cent. elongation. This decrease in the per cent. elongation is of vital interest to the rolling-mill man, it being evident that a piece of the annealed metal having the maxi-

mum per cent. elongation is in the best possible physical condition for withstanding further mechanical treatment.

The question will probably arise as to what sort of trouble may be brought about by overheating the copper in the annealing furnace. We have seen that overheating facilitates the development of polysynthetic twin crystals and results in a decrease in the per cent. elongation. Sheets which have been overheated in this way give trouble by cracking at the edges. If the annealing temperature is very high, gases which were unable to escape when the metal was cast tend to segregate at the granular boundaries. This results in blisters, which may make themselves evi-

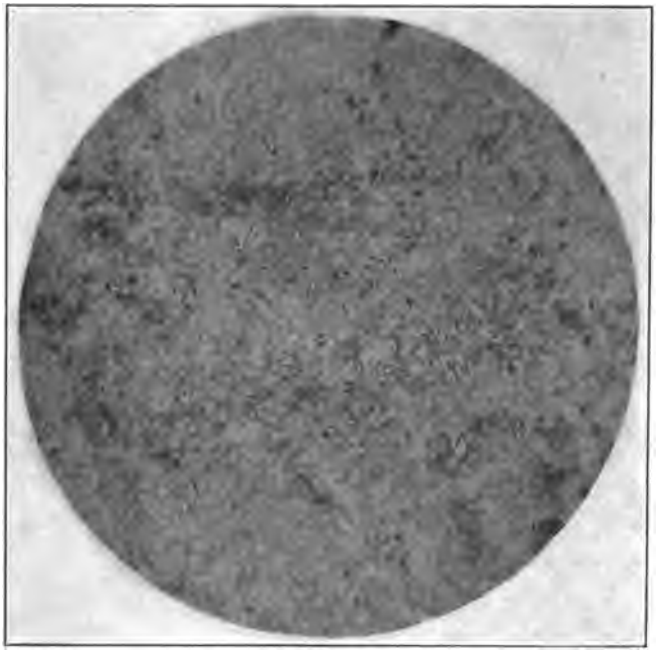

FIG. 10.—AFTER ANNEALING AT 932° F. × 100.

dent only upon subsequent rolling. Overheating of the copper previous to the breaking-down rolls has a similar effect. When the annealing temperature becomes so high that the metal begins to soften, the gas contained in the metal will tend to force the grains apart, causing cracks, into which the oxygen of the atmosphere will penetrate. The copper in this condition is said to be burnt and must be remelted. Overheating does not necessarily mean that the copper is burnt. If the copper attains in the annealing furnace a temperature in excess of 1,100° or 1,200° F. it may be said to have been overheated, but no harm necessarily results if the overheating is inconsiderable. We have already seen that copper

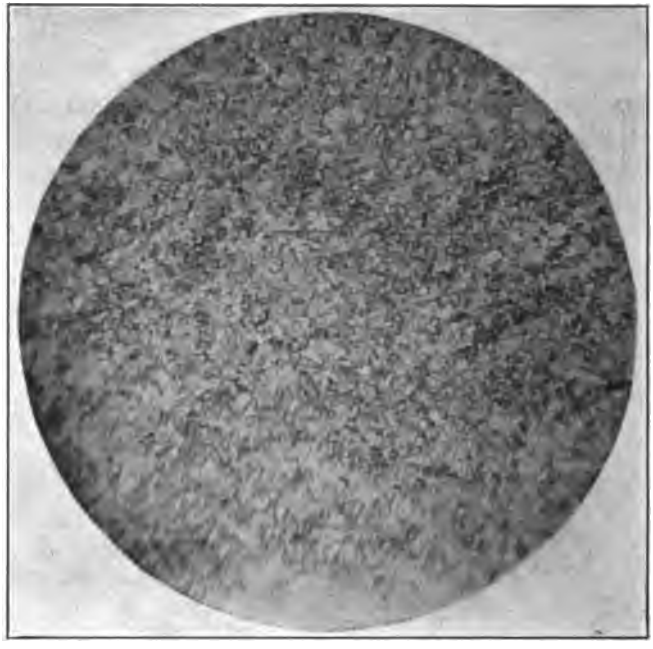

FIG. 11.—AFTER ANNEALING AT 1,112° F. × 100.

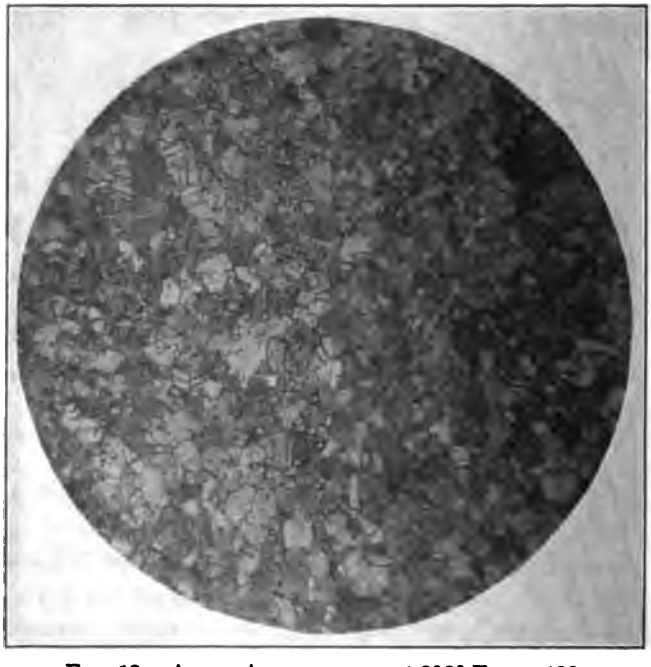

FIG. 12.—AFTER ANNEALING AT 1,292° F. × 100.

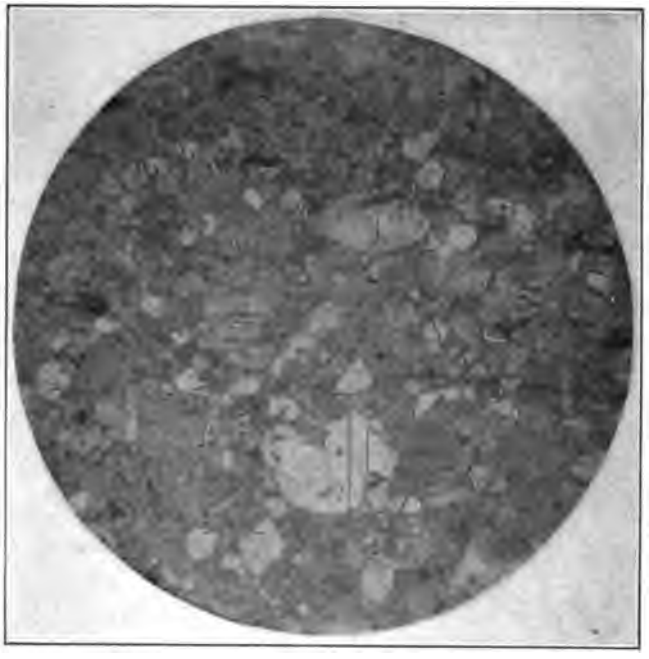

FIG. 13.—AFTER ANNEALING AT 1,472° F. × 100.

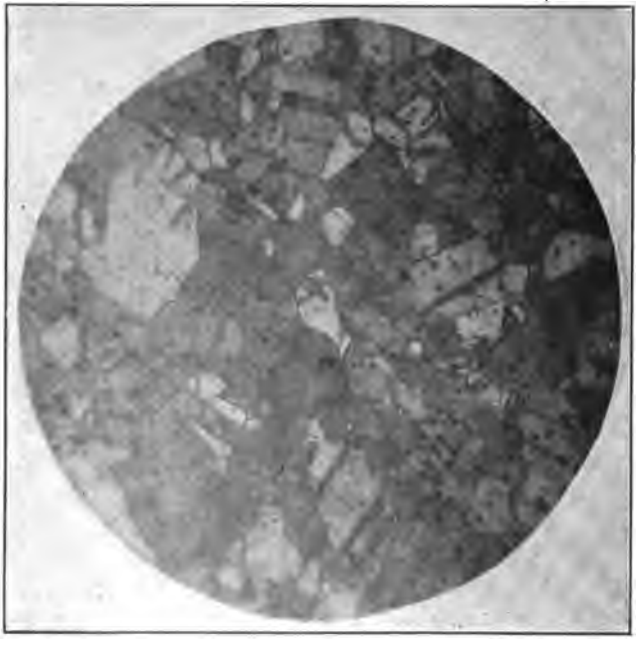

FIG. 14.—AFTER ANNEALING AT 1,652° F. × 100.

FIG. 15.—AFTER ANNEALING AT 1,832° F. × 100.

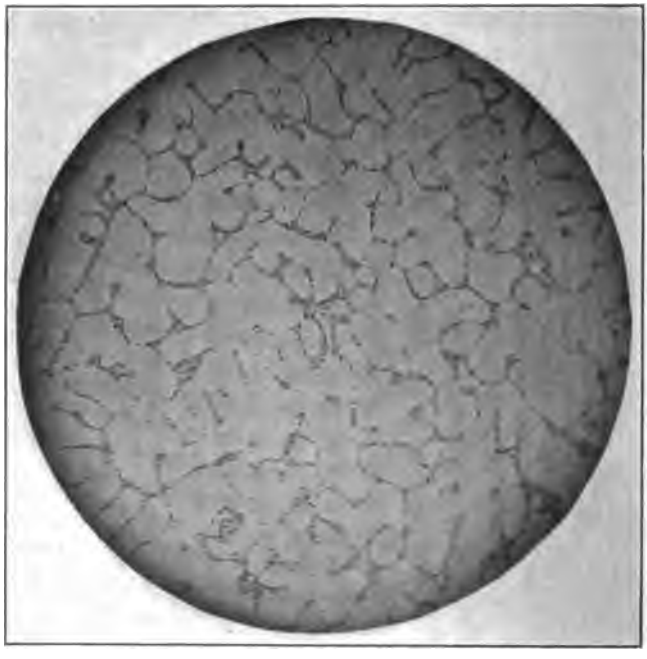

FIG. 16.—CAKE CONTAINING 0.076 PER CENT. OXYGEN. × 41.

low in oxygen will apparently permit of being heated to 1,300° to 1,400° F. before showing any considerable decrease in either the per cent. conductivity or per cent. elongation.

In order to further illustrate my point it may be of interest to show what happens when a cake of electrolytic copper is rolled into sheet. Fig. 16 is a photomicrograph of a piece of copper cut from a 14 by 17 in. 200-lb. cake containing 0.076 per cent. oxygen. In the photomicrograph the dark network is the eutectic mixture of Cu_2O and copper containing approximately 0.39 per cent. oxygen. The cake was heated to about

FIG. 17.—3/16 IN. SHEET ROLLED FROM CAKE CONTAINING 0.076 PER CENT. OXYGEN BEFORE SCALING. × 87.

1,560° F. and broken down to $\frac{3}{16}$ in. in thickness by repeated passes through the rolls. Fig. 17 gives a photomicrograph of a piece of the resulting sheet previous to its being scaled. In Fig. 17 it will be noted that while the eutectic has ceased to exist as a continuous network it has not been entirely broken up. A determination of the oxygen made by measuring the area occupied by the dark patches of eutectic and multiplying this area expressed in per cent. by 0.39 gives 0.077 as the oxygen content of the sheet, checking almost exactly the measurement previously made upon a piece of the copper cut from the cake before it was rolled. Note also the small crystals. The copper as it came from the rolls was at a temperature of about 1,100° F., a temperature sufficiently high to anneal

the sheet. In fact, a piece of this sheet was afterward rolled to 0.03 in. in thickness without further annealing. After trimming and cutting to suitable size for further rolling the $\frac{3}{16}$-in. sheet was scaled and pickled. Fig. 18 is a photomicrograph of the sheet after scaling and pickling, the magnification being the same as in the case of Fig. 17. The specimen was etched more particularly to show the eutectic areas. It will at once be seen that the structure has become very much coarser, which is due to the sheet having been heated to 1,450° F. when it was scaled. The

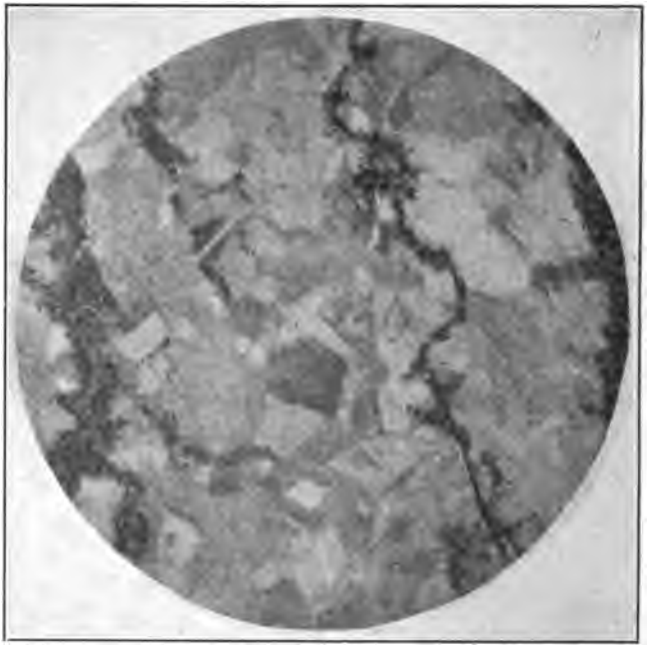

FIG. 18.—SHEET SHOWN IN FIG. 17 AFTER BEING SCALED. × 87.

eutectic has become very much segregated and the crystals are much larger. The crystalline structure is brought out more clearly in Fig. 19. The polysynthetic crystals mentioned in a preceding paragraph are in evidence. The $\frac{3}{16}$-in. sheet was now cold rolled to 0.033 in. The sheet rolled fairly well but cracked at the edges. Fig. 6 shows the structure of the sheet after being rolled to 0.033 in. After being rolled the sheet was annealed at a temperature of about 1,200° F. Fig. 20 is a photomicrograph of the annealed sheet. It will be seen that the eutectic structure has been entirely broken up. The photomicrograph shows a nearly complete absence of twin crystals, indicating that the annealing temperature was about right; in fact, the sheet was subsequently rolled to 0.005 in. gauge without developing defects of any sort. The presence of twin crystals in Fig. 19 shows that the sheet was overheated during

the scaling process. Had the temperature been kept at about 1,100° F. instead of being allowed to reach 1,450° F. or thereabouts the metal would have been in much better physical condition for the subsequent reduction to 0.033 in. and the sheets would not have cracked at the edges.

The use of pyrometers for controlling the annealing furnace cannot be too strongly urged. In the ordinary reverberatory annealing furnace means should be provided for determining the temperature at several points in the furnace. Careful watch should be kept to see that sheets

FIG. 19.—SAME AS FIG. 18, BUT ETCHED TO SHOW POLYSYNTHETIC TWIN CRYSTALS × 87.

nearest the bridge wall are not being overheated. Pyrometers to be of use in controlling the annealing process should be frequently checked against a standard thermo-couple used for this purpose only, otherwise the readings may become misleading. Copper at 1,100° F. is at a dull red heat. If one looks into the furnace the sheet when at this temperature appears almost black, the reddish glow being only faintly visible. A full dull red indicates a temperature of 1,300° F., and a dull cherry-red, which I have observed in many annealing furnaces, indicates a temperature of 1,475° F. Optical pyrometers are very nearly useless for measuring temperatures as low as 1,100° to 1,200° F. Thermo-electric pyrometers unless frequently checked may give readings that are 150° to 200° too low.

In these experiments the time factor has been neglected; not, however, because it has not been considered as of importance. Baucke[2] has shown that prolonged heating up to 450° C. yields the same results as heating at 1,025° C. for 10 min., and that heating at even as low a temperature as 100° C. for 350 days will result in incipient recrystallization. Results which might be obtained in the laboratory would not be applicable directly in practice. This much, however, may be said: exposure to an annealing temperature of 1,100° F. for 20 min. is sufficient to perfectly

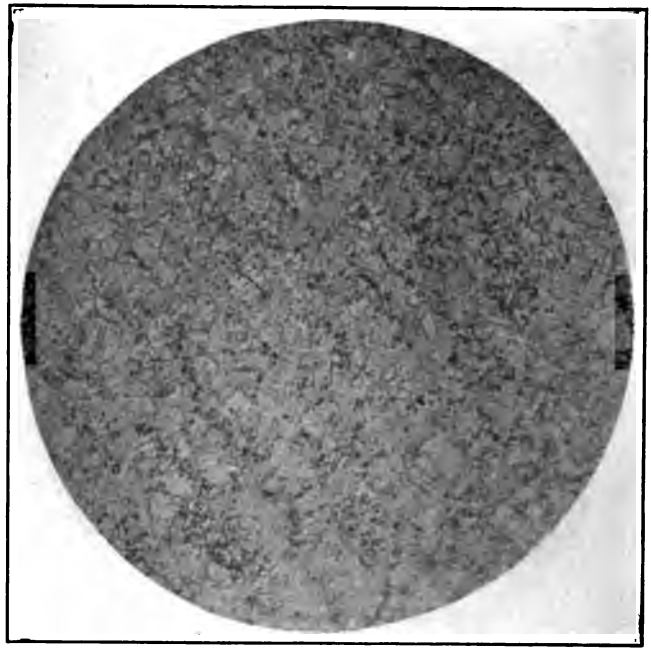

FIG. 20.—SHEET COPPER AFTER BEING COLD ROLLED TO 0.033 IN. AND ANNEALED. × 87.

anneal cold-rolled copper of any gauge up to $\frac{3}{16}$ in., it being understood that the copper shall attain this temperature and be held at this temperature for the stated period; and further, that exposure to this temperature for a period of an hour does not injure the copper. Where annealing is carried out at temperatures higher than 1,100° the time factor has a far greater effect.

I am well aware that it is impossible in a rolling mill to carry out the operation of annealing with the degree of perfection attainable in a laboratory experiment. I am, however, of the opinion that the experiments here described may be suggestive and helpful to those actively engaged in the operation of copper rolling mills.

[2] H. Baucke, Amsterdam: On Some Recent Micrographical Investigations of Copper, International Association for Testing Materials, vol. ii, No. 11, II_{14}.

Discussion

Lawrence Addicks, Chrome, N. J.—Eleven years ago I presented a paper before the Institute of Electrical Engineers covering this same field, but less comprehensively than it has been presented here, and at that time I made a study of the correspondence between tensile strength and conductivity, and found a straight-line relation between the two as you soften a wire by annealing, or harden it by drawing. Recalling this, there is one thing I want to ask about, which I think must be a misprint. I notice on p. 759 that at the 700° temperature there was obtained a soft wire apparently without a rise in conductivity.

I might also add that a great deal of unpublished work along these lines has been carried out at some of the Eastern brass works, and that it has been found that the temperature of annealing is affected by impurities. Also that the time of exposure plays a part.

Curves for the Sensible-Heat Capacity of Furnace Gases

BY C. R. KUZELL AND G. H. WIGTON, GREAT FALLS, MONT.

(Salt Lake Meeting, August, 1914)

INTRODUCTION

KNOWLEDGE of the thermal capacity of gases is of great importance in making metallurgical calculations.

The metallurgist is frequently called upon to investigate and determine furnace efficiencies in which the heat carried into or out of the furnace by gases is a large item in the heat balance. Not only do such problems present themselves in the determination of furnace efficiency, but also in the study of the application of heat in accessory apparatus, such as stoves, regenerators, waste-heat boilers, driers, etc. The thermal effect of the use of excess air in the combustion of fuel, the theoretical temperatures of combustion, the quantity of heat in hot blast at various temperatures, the effect of hot blast on furnace temperatures, are a few more examples of frequently occurring calorific problems involving gases. So many are the applications of the data on the heat capacity of gases that the subject merits careful study.

The heat in a gas may be due to its heat of combustion, if it is a combustible gas, and to its temperature. The latter is called its sensible heat and is the heat absorbed or evolved by a gas as its temperature is raised or lowered. The heat of combustion of gases is well established and is commonly known. The values can be found in almost any modern treatise on metallurgy. The sensible heat of gases has not been so well established because it is in most cases a function of the temperature, and the values of specific heats of gases over a wide range of temperatures have only recently been determined. It is the purpose of this paper to deal with the sensible heat of gases.

The calculation of sensible-heat capacities[1] from specific-heat equations is a comparatively long and tedious operation. This is especially

[1] Dr. Fulton uses the term "thermal capacity" (*Principles of Metallurgy*, p. 410). However, in the case of a combustible gas, the term "thermal capacity" might be erroneously supposed by some to include heat of combustion.

true in the case of polyatomic gases the specific-heat equations of which are quadratics or are higher in degree.

To simplify this calculation the authors have selected what have appeared to be the most reliable and recent data and have constructed a series of curves by means of which the sensible heat in a gas at any temperature or between any temperatures up to 2,000° C., or 3,600° F., may be easily determined. The use of these curves has not only effected a great saving in time, but has encouraged the making of many calculations which would not have been attempted if the more laborious method was used.

Experimental Data on the Specific Heat of Gases

Dr. Joseph W. Richards has, in his *Metallurgical Calculations*, Part I, given formulæ for the mean specific heats of all the common gases in terms of both the imperial and metric systems of measurement. These formulæ are derived from the results of experimental determination of the specific heats at various temperatures. The data were obtained from the researches of Regnault, Mallard, and Le Chatelier. Since the time of publication of that book, however, other investigators have been at work on the subject. Among these are: Pier, Holborn, Austin, Henning, Njerrum, Furstenau, and Swan. A summary of the results of their researches has been very ably made by Messrs. Lewis and Randall, who presented their conclusions in a paper entitled A Summary of the Specific Heats of Gases.[2] They have selected formulæ for molal heats[3] of gases to agree as closely as possible with all the experimental work, and where a difference in data existed the values obtained by the most reliable method were chosen. Since their formulæ agree with the most probable values of all the experimental data, they are especially valuable, and have been used in constructing the curves for heat capacity that are the subject of this paper.

Specific-Heat Equations and Their Derivation

As noted above, Lewis and Randall give formulæ for the molal heats of the gases. It has been necessary to convert these formulæ into more convenient form for use with either volumetric or gravimetric units in either the Imperial or Metric system.

To facilitate the use of formulæ, the following symbols have been adopted:

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[2] See *Journal of the American Chemical Society*, vol. xxxiv, No. 9.

[3] The molal heat is the product of the specific heat and the molecular weight.

C_p = molal heat = heat required to raise the temperature of a molecular weight of the gas 1° under constant pressure.[4]
c_p = specific heat under constant pressure.
$c_m(t_1 \text{ to } t)$ = mean specific heat between the temperatures t_1 and t.
T = absolute temperature, degrees centigrade.
t = temperature, degrees centigrade.
t = temperature, degrees Fahrenheit.
$Q(t_1 \text{ to } t)$ = quantity of sensible heat (heat capacity) between the temperatures t_1 and t.

The following formulæ were selected by Lewis and Randall:

Nitrogen, oxygen, carbon monoxide,	$C_p = 6.50 + 0.0010T$
Hydrogen,	$C_p = 6.50 + 0.0009T$
Water vapor, hydrogen sulphide,	$C_p = 8.81 - 0.0019T + 0.00000222T^2$
Carbon dioxide, sulphur dioxide,	$C_p = 7.0 + 0.0071T - 0.00000186T^2$

Table I gives the formulæ for mean specific heats that we have calculated from the above molal heats for 1 kg., 1 cu. m., 1 lb., and 1 cu. ft. of each gas.

The values for methane are calculated from a formula given by Fulton (*Principles of Metallurgy*, p. 408) on the authority of J. V. Ehrenwerth (*Metallurgie*, vol. vi, p. 306).

For an explanation of the method of converting specific-heat formulæ to mean specific-heat formulæ see Appendix A.

Heat-Capacity Curves; Construction and General Application

The amount of sensible heat that a gas will absorb between any two temperatures t_1 and t is equal to the product of the mean specific heat between those two temperatures times the difference in temperature. Therefore, if one temperature is zero,

$$Q(0 \text{ to } t) = c_m\, t$$

By substituting in this equation the various values of c_m given in the preceding tabulation and then substituting various values of t, any desired number of values of Q may be obtained. From these values a curve can easily be plotted. Such a curve will give for any temperature up to 2,000° C., or 3,600° F., the sensible-heat capacity of the gas.

On Plate I we have drawn curves for each gas giving the sensible-heat capacity in British thermal units per pound and also per cubic foot (measured under standard conditions). On Plate II are corresponding

[4] Gases in metallurgical furnaces usually expand under constant pressure. If the gas is confined to a constant volume, external work is done to maintain the constancy of volume. In such a case the amount of the outer work may be calculated (see Richards's *Metallurgical Calculations*, pt. 1, p. 60) by subtracting from the specific heat at constant pressure 2 calories per degree for a molecular weight, or 0.09 calorie per degree for 1 cu. m., or 0.09 ÷ weight of 1 cu. m. for a kilogram of gas.

TABLE I.—*Mean Specific Heats of Gases Under Constant Pressure up to 2,000° C., or 3,600° F.*
($t = °C.$, $t = °F.$)

	N_2	O_2
c_m(0 to t) for 1 kg. in kg-cal	$0.2417 + 0.0000178t$	$0.2117 + 0.0000156t$
c_m(0 to t) for 1 cu. m. in kg-cal	$0.3025 + 0.0000223t$	$0.3025 + 0.0000223t$
c_m(0 to t) for 1 lb. in B.t.u.	$0.2411 + 0.0000099t$	$0.2111 + 0.0000087t$
c_m(0 to t) for 1 cu. ft. in B.t.u.	$0.0188 + 0.0000008t$	$0.0188 + 0.0000008t$

	H_2O	H_2S
c_m(0 to t) for 1 kg. in kg-cal	$0.4694 - 0.0000191t + 0.0000000411t^2$	$0.2481 - 0.0000101t + 0.0000000217t^2$
c_m(0 to t) for 1 cu. m. in kg-cal	$0.3777 - 0.0000154t + 0.0000000330t^2$	$0.3777 - 0.0000154t + 0.0000000330t^2$
c_m(0 to t) for 1 lb. in B.t.u.	$0.4701 - 0.0000118t + 0.0000000127t^2$	$0.2485 - 0.0000062t + 0.0000000067t^2$
c_m(0 to t) for 1 cu. ft. in B.t.u.	$0.0236 - 0.0000006t + 0.0000000006t^2$	$0.0236 - 0.0000006t + 0.0000000006t^2$

	Air	CO	H_2
c_m(0 to t) for 1 kg. in kg-cal	$0.2348 + 0.0000173t$	$0.2419 + 0.0000179t$	$3.3461 + 0.0002232t$
c_m(0 to t) for 1 cu. m. in kg-cal	$0.3025 + 0.0000223t$	$0.3025 + 0.0000223t$	$0.2995 + 0.0000200t$
c_m(0 to t) for 1 lb. in B.t.u.	$0.2342 + 0.0000096t$	$0.2413 + 0.0000099t$	$3.3381 + 0.0001240t$
c_m(0 to t) for 1 cu. ft. in B.t.u.	$0.0188 + 0.0000008t$	$0.0188 + 0.0000008t$	$0.0187 + 0.0000007t$

	CO_2	SO_2	CH_4
c_m(0 to t) for 1 kg. in kg-cal	$0.2000 + 0.0000691t - 0.0000000141t^2$	$0.1373 + 0.0000475t - 0.0000000097t^2$	$0.593 + 0.0000741t$
c_m(0 to t) for 1 cu. m. in kg-cal	$0.3930 + 0.0001359t - 0.0000000277t^2$	$0.3930 + 0.0001359t - 0.0000000277t^2$	$0.425 + 0.0000530t$
c_m(0 to t) for 1 lb. in B.t.u.	$0.1975 + 0.0000388t - 0.0000000043t^2$	$0.1356 + 0.0000267t - 0.0000000030t^2$	$0.5904 + 0.0000411t$

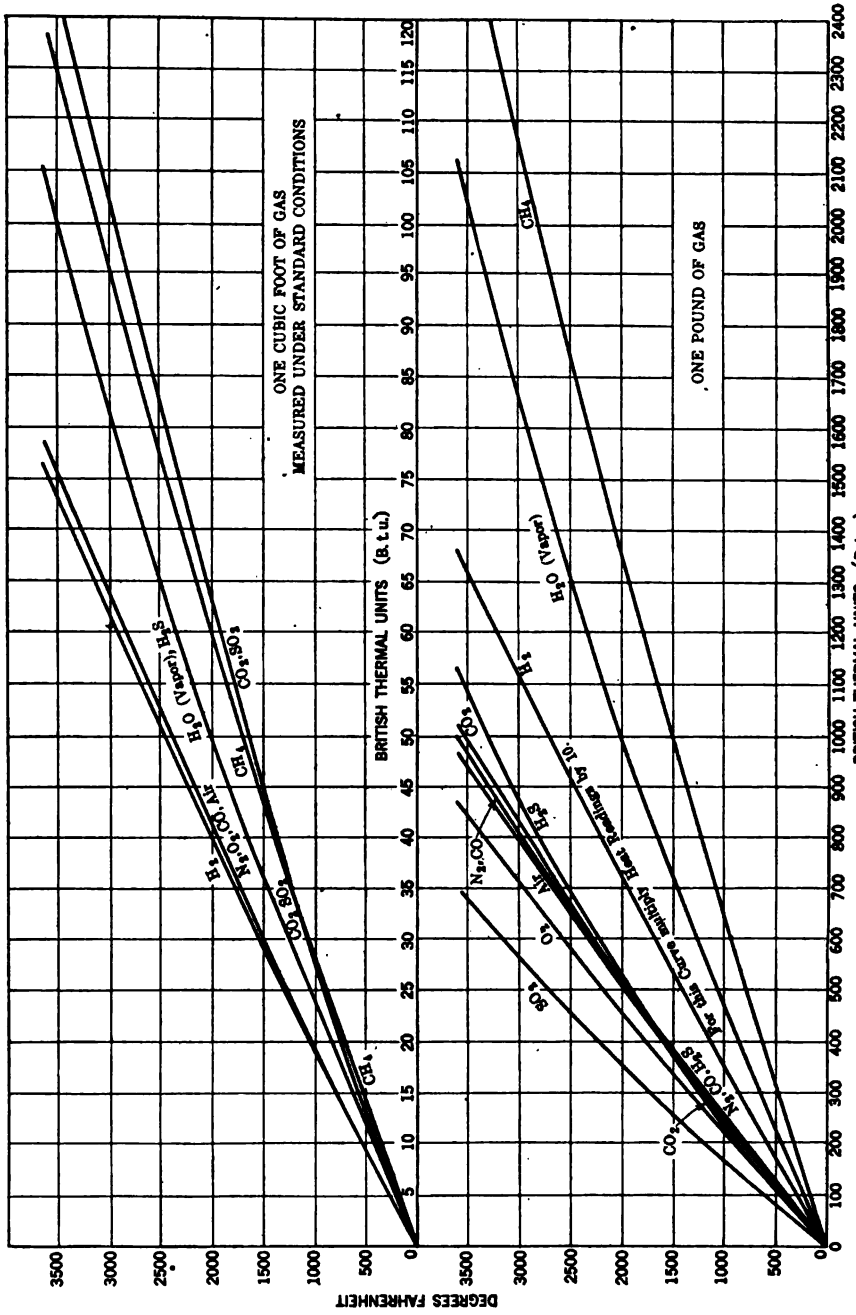

PLATE I.—SENSIBLE-HEAT CAPACITY FROM ZERO DEGREES FAHRENHEIT TO VARIOUS TEMPERATURES. EXPANSION UNDER CONSTANT PRESSURE.

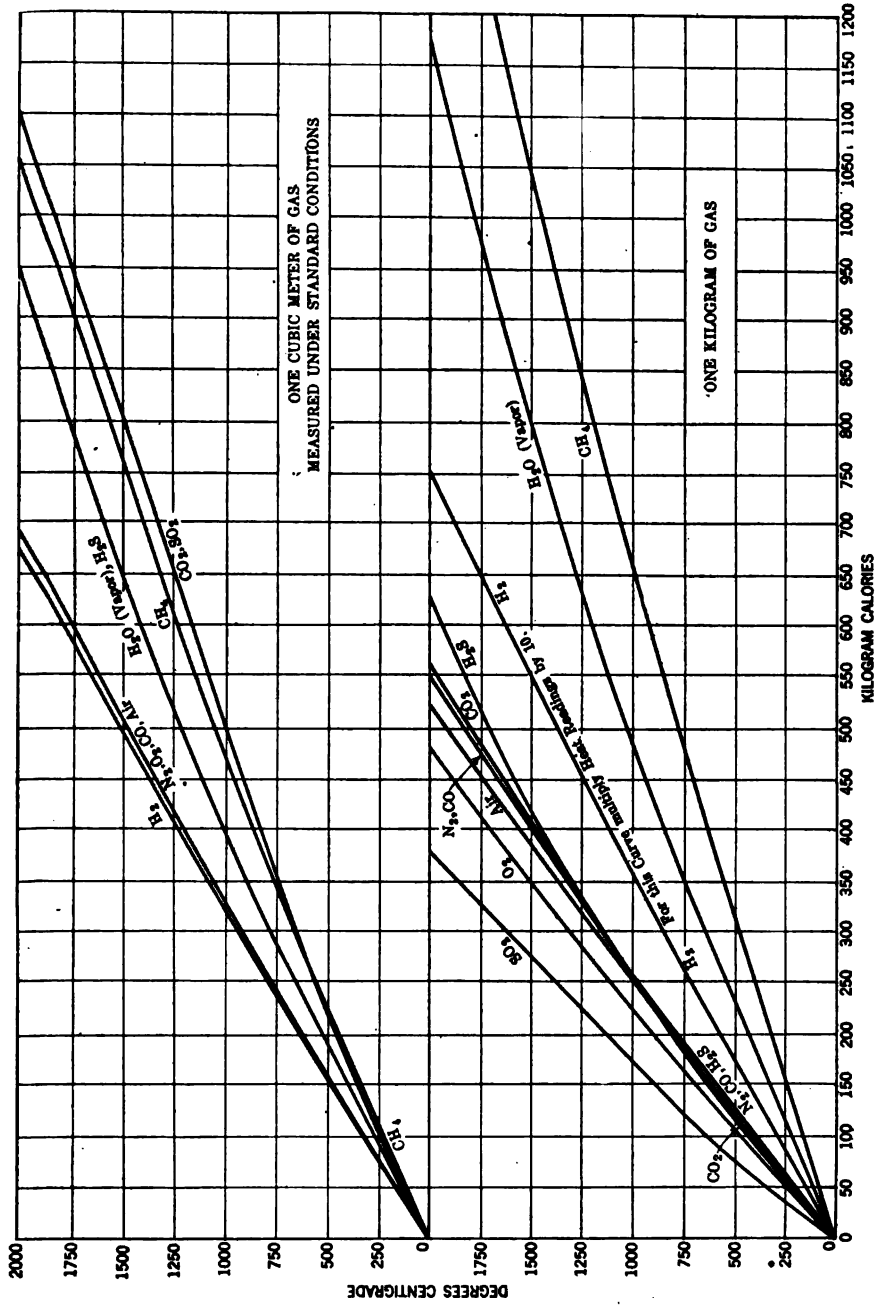

PLATE II.—SENSIBLE-HEAT CAPACITY FROM ZERO DEGREES CENTIGRADE TO VARIOUS TEMPERATURES. EXPANSION UNDER CONSTANT PRESSURE.

curves giving kilogram-calories per kilogram and per cubic meter (also measured under standard conditions).

It should be noted that the values given apply to the substances only in the gaseous state.

Dissociation is not great within the range of temperatures for which the curves are drawn. (See Fulton, *Principles of Metallurgy*, p. 414.)

The general application of the curves is simple. The temperatures are laid off along the axis of ordinates and the sensible-heat capacities are laid off along the axis of abscissæ. If the temperature of a gas is known and it is desired to find the sensible-heat capacity between zero and this temperature, one reading gives it directly. If two temperatures are given and the sensible-heat capacity between these two temperatures is desired, two readings must be taken and subtracted one from the other.

In addition to Plate I and Plate II we have drawn two more series of curves in Plates III and IV, which give the actual specific heats at various temperatures, the former for units of weight and the latter for units of volume. These are for use in finding the theoretical temperature of combustion, as explained in Appendix B and as illustrated in Example 4.

Application of Heat-Capacity Curves in Metallurgical Calculations

The applications of the curves to metallurgical problems are many. On account of lack of space only four typical examples, just enough to illustrate the use of the curves, will be worked out. Special attention is directed to Example 4, which illustrates a method of finding the theoretical temperature of combustion much more easily than by any other method we know of.

Example 1

A reverberatory furnace discharges 10,000 cu. ft. of waste gases per minute (measured under standard conditions) at 2,000° F. The volumetric analysis of the gas is as follows:

H_2O	$CO_2 + SO_2$	O_2	N_2
8	15	2	75

Find the sensible heat in the waste gases above 0°F.

Referring to Plate I, we obtain the following values for each of the constituent gases at 2,000° F.

Sensible-heat capacity, B.t.u. per cubic foot.

H_2O	$CO_2 + SO_2$	$O_2 + N_2$
49.6	63.6	40.8

Combining these in the proper proportion we get:

$94.6 \times 0.08 + 63.6 \times 0.15 + 40.8 \times 0.77 = 44.92$ B.t.u. per cubic foot,

which is the sensible heat in 1 cu. ft. of the waste gases. Therefore the total sensible heat passing out of the furnace in the waste gases each minute is 449,200 B.t.u.

Example 2

Suppose the gases of Example 1 enter a brick regenerator at 2,000° F. and leave it at 600° F., and that the regenerator subsequently heats 8,000 cu. ft. of air from 70° F. to 1,200° F. for every 10,000 cu. ft. of waste gases (both air and gases being measured under standard conditions). What is the calorific efficiency of the regenerator? From Example 1 we know that the sensible heat in the gases at 2,000° F. is 449,200 B.t.u. Proceeding in the same manner (using Plate I), we find that at 600° F. the sensible heat per cubic foot is:

$$14.1 \times 0.08 + 16.2 \times 0.15 + 11.6 \times 0.77 = 12.49 \text{ B.t.u.}$$

The sensible heat in 10,000 cu. ft. at 600° is therefore 124,900 B.t.u. The heat input of the regenerator is therefore

$$449{,}200 - 124{,}900 = 324{,}300 \text{ B.t.u.}$$

The heat in 1 cu. ft. of air at 1,200° F. is 23.7 B.t.u. and at 70° F. it is 1.3 B.t.u. Therefore, the net heat absorbed by 1 cu. ft. of air is 22.4 B.t.u. and the heat output of the regenerator is 8,000 × 22.4, or 179,200 B.t.u.
∴ Efficiency of regenerator =

$$\frac{\text{Heat output}}{\text{Heat input}} \times 100 = \frac{179{,}200 \times 100}{324{,}300} = 55.3 \text{ per cent.}$$

Example 3

At an iron blast-furnace plant a stove receives 35,000 cu. ft. of blast-furnace gas per minute. The heating value of this gas is 98 B.t.u. per cubic foot and its volumetric composition is:

CO_2	CO	H_2	CH_4	H_2O	O_2	N_2
14.0	24.0	3.5	0.5	3.0	0.0	55.0

The gas is at a temperature of 600° F.

The stove discharges 61,000 cu. ft. of gases at 550° F. and of the following volumetric composition.

CO_2	H_2O	O_2	N_2
22.1	4.3	2.0	71.6

The temperature of the air that is used for burning the gas in the stove is 50° F.

The stove (while on air) heats 100,000 cu. ft. of air from 50° F. to 1,100° F. for every 35,000 cu. ft. of the original blast-furnace gas.

All volumes are measured under standard conditions of temperature and pressure.

What is the efficiency of the stove?

The per cent. efficiency is $\frac{\text{Heat output}}{\text{Heat input (net)}} \times 100.$

	B.t.u.
Heat input:	
Heat of combustion of 35,000 cu. ft. at 98 B.t.u. =	3,330,000
(From Plate I) Sensible heat in 35,000 cu. ft. gas at 600° F. $35{,}000 \times [16.2 \times 0.14 + 11.6 \times (0.24 + 0.035 + 0.55) + 14.1 \times 0.03 + 17.0 \times 0.005]$ =	632,250
(From Plate I) Since the air required for combustion is 30,700 cu. ft. (by calculation), the sensible heat in it at 50° F. is $30{,}700 \times 0.9$ =	27,630
Gross heat input of stove,	3,989,880
Deduct sensible heat in 61,000 cu. ft. gases at 550° F. $61{,}000 \times [14.7 \times 0.221 + 12.9 \times 0.043 + 10.5 \times 0.736]$ =	703,330
Net heat input of stove,	3,286,550
Heat output:	
(From Plate I) Sensible heat in 100,000 cu. ft. air at 1,100° F. $100{,}000 \times 21.6$ =	2,160,000
(From Plate I) Sensible heat in 100,000 cu. ft. air at 50° F. $100{,}000 \times 0.9$ =	90,000
Heat output of stove =	2,070,000

Efficiency of stove:

$$\text{Per cent. efficiency} = \frac{2{,}070{,}000}{3{,}286{,}550} = 63 \text{ per cent.}$$

Example 4

Find the theoretical temperature of combustion, or "pyrometric effect," of the blast-furnace gas in *Example* 3.

The total sensible-heat capacity of the 61,000 cu. ft. of the products of combustion is 3,989,880 B.t.u., or 65.4 B.t.u. per cubic foot

The composition of the products of combustion is:

CO_2	H_2O	O_2	N_2
22.1	4.3	2.0	71.6

The theoretical temperature of combustion t is found by the formula:

$$t = \frac{t_1p_1c_1 + t_2p_2c_2 + t_3p_3c_3 + \text{etc.}}{p_1c_1 + p_2c_2 + p_3c_3 + \text{etc.}}$$

in which, t_1, t_2, t_3, etc., are the theoretical temperatures of combustion of each of the constituent gases, if we consider for a moment that the gases contain only that constituent and no other. In other words t_1, t_2, t_3, etc., are the temperatures corresponding to the points of intersection of the curves on Plate I with an ordinate drawn through 65.4 B.t.u.

p_1, p_2, p_3, etc., are equal to the volumetric percentage of each constituent gas.

c_1, c_2, c_3, etc., are equal to the mean specific heat of 1 cu. ft. of each constituent gas between t and t_1, t and t_2, etc., and are obtained by reading the specific heats from the curves of Plate III just above or below the temperatures t_1, t_2, t_3, etc., so that the readings will be very closely the mean specific heats between t and t_1, t and t_2, etc.

For a full explanation of the method of obtaining the formula for the theoretical temperature see Appendix B.

In this particular example we shall let the subscripts 1, 2, 3, and 4 represent respectively, CO_2, H_2O, O_2, and N_2.

From Plate I, we obtain by erecting an ordinate through 65.4 B.t.u.

$$t_1 = 2{,}045° \text{ F.}, \quad t_2 = 2{,}520° \text{ F.}, \quad t_3 \text{ and } t_4 = 3{,}075° \text{ F.}$$

From Plate III, we obtain:

$$c_1 = 0.0385, \quad c_2 = 0.0335, \quad c_3 \text{ and } c_4 = 0.0235$$

From the analysis:

$$p_1 = 22.1, \quad p_2 = 4.3, \quad p_3 = 2.0, \quad p_4 = 71.6$$

Substituting: t =

$$\frac{(2{,}045 \times 22.1 \times 0.0385) + (2{,}520 \times 4.3 \times 0.0335) + (2.0 + 71.6)\ (3{,}075 \times 0.0235)}{(22.1 \times 0.0385) + (4.3 \times 0.0335) + (73.6 \times 0.0235)}$$

$$= \frac{7{,}423}{2.725} = 2{,}724°\text{F.}$$

Appendix A

Method of Transforming the Equation for Specific Heat into the Equation for Mean Specific Heat

Let the curve in Fig. 1 represent the general equation for specific heat,

$$c_p = A + BT + CT^2 + DT^3 + \text{etc.}$$

In which A, B, C, D, etc., are constants and T is the temperature.

Let T_1 and T_2 be any two temperatures on the curve.

The mean specific heat c_m between T_1 and T_2 will therefore be,

$$c_m(T_1 \text{ to } T_2) = \frac{c_1 dT_1 + c_2 dT_2 + c_3 dT_3 + \ldots + c_n dT_n}{dT_1 + dT_2 + dT_3 + \ldots + dT_n}$$

As dT decreases without limit,

$$c_m(T_1 \text{ to } T_2) = \frac{\int_{T_1}^{T_2} c\,dT}{\int_{T_1}^{T_2} dT} = \frac{\int_{T_1}^{T_2} c\,dT}{T_2 - T_1}$$

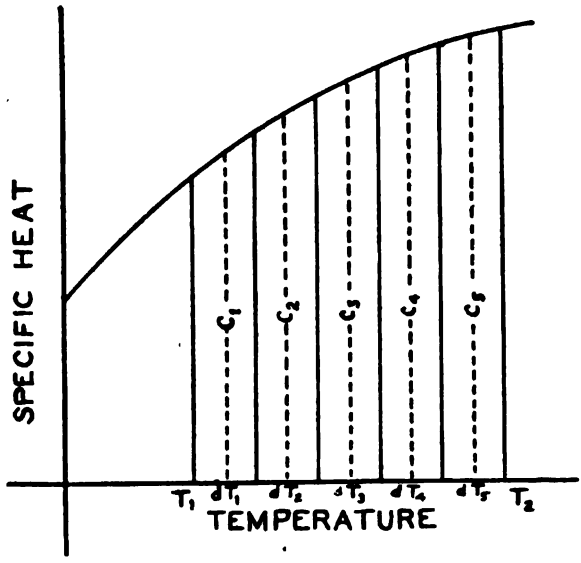
SPECIFIC HEAT
C_1
C_2
C_3
C_4
C_5
T_1
dT_1
dT_2
dT_3
dT_4
dT_5
T_2
TEMPERATURE

FIG. 1.

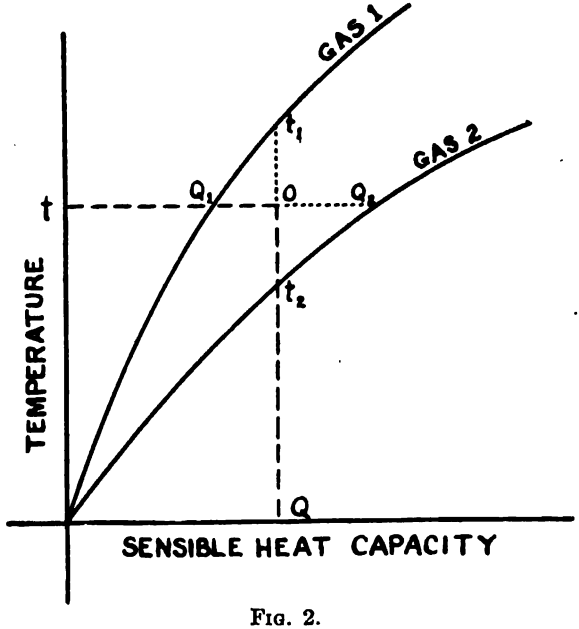
GAS 1
GAS 2
t_1
t_2
t
Q_1
O
Q_2
Q
TEMPERATURE
SENSIBLE HEAT CAPACITY

FIG. 2.

Substituting the general equation for c,

$$c_m(T_1 \text{ to } T_2) = \frac{\int_{T_1}^{T_2} (A + BT + CT^2 + DT^3 + \text{etc.})dT}{T_2 - T_1}$$

$$= \frac{A(T_2 - T_1) + \frac{B}{2}(T_2^2 - T_1^2) + \frac{C}{3}(T_2^3 - T_1^3) + \text{etc.}}{T_2 - T_1}$$

$$= A + \frac{B}{2}(T_2 + T_1) + \frac{C}{3}(T_2^2 + T_1T_2 + T_1^2) + \text{etc.}$$

When $T_1 = 0$,

$$c_m(0 \text{ to } T) = A + \frac{B}{2}T + \frac{C}{3}T^2 + \text{etc.}$$

Appendix B

Derivation of Formula for Finding the Theoretical Temperature of Combustion

Consider the products of combustion of a combustible gas as made up of several constituent gases, which we shall call Gas 1, Gas 2, Gas 3, etc.

In Fig. 2, the curves represent the sensible-heat capacity curves for the series of constituent gases, Gas 1, Gas 2, Gas 3, etc.

Q = sensible-heat capacity of the combined gases of combustion from 0° to t°.

Q_1, Q_2, Q_3, etc. = ditto for Gases 1, 2, 3, etc., from 0° to t_1, t_2, t_3, etc.

t = theoretical temperature of combustion of the gas.

t_1, t_2, t_3, etc., = theoretical temperature of combustion if the total product of combustion were only one gas, Gas 1, Gas 2, etc.

p_1, p_2, p_3, etc. = portions of Gas 1, Gas 2, Gas 3, etc., in the products of combustion.

It is evident that $Q = p_1Q_1 + p_2Q_2 + p_3Q_3 + \text{etc.}$

From the figure $Q_1 = Q + \bar{O}Q_1$, and $Q_2 = Q + \bar{O}Q_2$, etc.

$$\therefore Q = p_1(Q + \bar{O}Q_1) + p_2(Q + \bar{O}Q_2) + \text{etc.}$$

whence,

$$Q = Q(p_1 + p_2 + \text{etc.}) + (p_1\bar{O}Q_1 + p_2\bar{O}Q_2 + \text{etc.})$$

Since

$$p_1 + p_2 + p_3 + \text{etc.} = 1$$

$$p_1\bar{O}Q_1 + p_2\bar{O}Q_2 + \text{etc.} = 0 \qquad (1)$$

It is evident that

$$t = t_1 + \bar{t_1}0 = t_2 + \bar{t_2}0 = \text{etc.} \qquad (2)$$

Let m_1, m_2, m_3 = average slopes of curves connecting the points Q_1 and t_1, Q_2 and t_2, etc.

Then

$$OQ_1 = \frac{t_1 0}{m_1}, \quad OQ_2 = \frac{t_2 0}{m_2}, \text{ etc.}$$

Substituting these values in equation (1), we get

$$\frac{p_1}{m_1} t_1 0 + \frac{p_2}{m_2} t_2 0 + \text{etc.} = 0 \tag{3}$$

From equations (2) and (3) we get the following simultaneous equations:

$$\frac{p_1}{m_1} t_1 0 + \frac{p_2}{m_2} t_2 0 + \text{etc.} = 0$$

$$t_1 - t_2 = t_2 0 - t_1 0$$

$$t_1 - t_3 = t_3 0 - t_1 0$$

Solving for $t_1 0$ and substituting in $t = t_1 0 + t_1$

$$t = \frac{\frac{p_1 t_1}{m_1} + \frac{p_2 t_2}{m_2} + \text{etc.}}{\frac{p_1}{m_1} + \frac{p_2}{m_2} + \text{etc.}}$$

Since the reciprocal of the slope is the specific heat and the reciprocal of the average slope is the mean specific heat, we have:

$$\frac{1}{m} = c$$

$$t = \frac{t_1 p_1 c_1 + t_2 p_2 c_2 + t_3 p_3 c_3 + \text{etc.}}{p_1 c_1 + p_2 c_2 + p_3 c_3 + \text{etc.}}$$

c_1, c_2, c_3, etc., may be obtained from Plate III or Plate IV, depending on whether p_1, p_2, p_3, etc., are volumetric or gravimetric percentages. Example 4 illustrates the procedure.

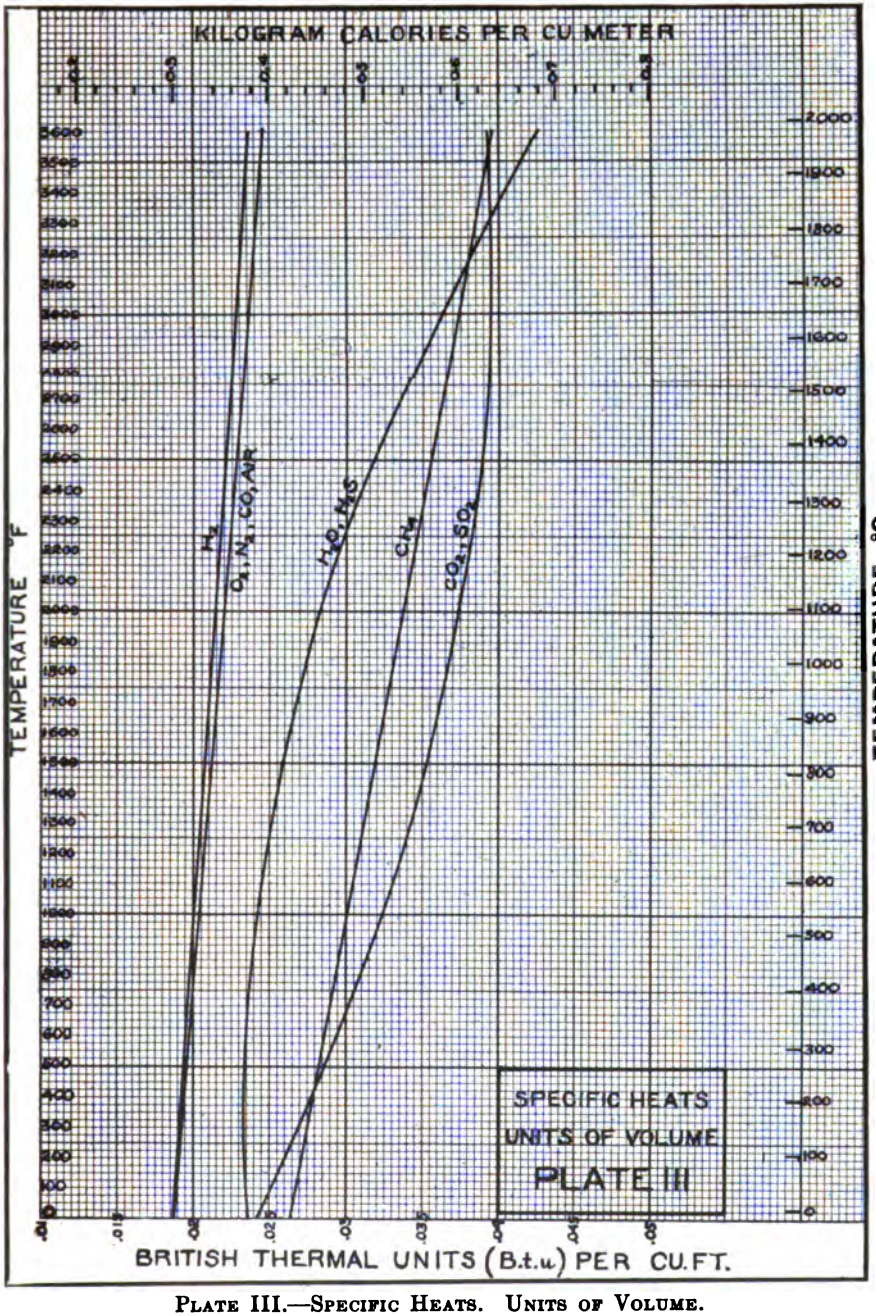

PLATE III.—SPECIFIC HEATS. UNITS OF VOLUME.

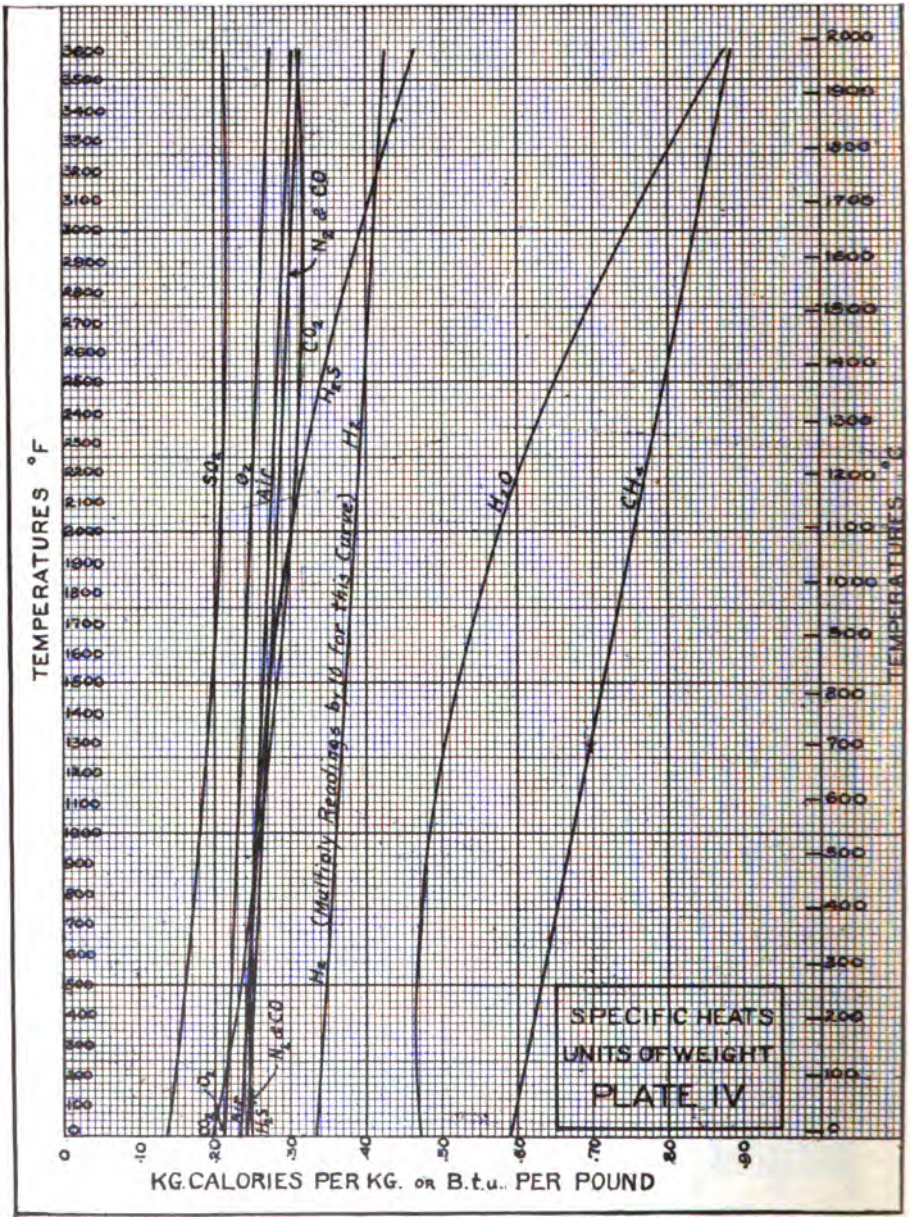

PLATE IV.—SPECIFIC HEATS. UNITS OF WEIGHT.

Losses of Zinc in Mining, Milling, and Smelting

BY DORSEY A. LYON AND SAMUEL S. ARENTZ, SALT LAKE CITY, UTAH

(Salt Lake Meeting, August, 1914)

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INTRODUCTION

A GREAT DEAL of attention has recently been given to the metallurgy of zinc, and much of that which has been said and written on this subject has been in the nature of a criticism. For instance, we are quite accustomed to hearing it said that the metallurgy of zinc has stood still, that it is antiquated, out of date, etc. We are, however, inclined to agree with George C. Stone[1] when he says that, although there have been no such spectacular and radical changes as were worked in the metallurgy of copper and iron by the introduction of converters, there has nevertheless been a decided and marked progress in the metallurgy of zinc, and especially during recent years. However, it must be admitted that, due to the fact that the chemistry of zinc is so different from that of the other metals, the retort process is the only process by which zinc is produced commercially; and, moreover, that losses which occur in the zinc industry—that is, in the mining, milling, and smelting of zinc ore—are very great, for it is stated[2] that probably less than 50 per cent. of the zinc mined, even

[1] The Metallurgy of Zinc. A Discussion. *Bulletin No.* 85, Jan., 1914, p. 163.

[2] Parsons, C. L.: Notes on Mineral Wastes. *Bulletin No.* 47, *U. S. Bureau of Mines* (1912).

TABLE I.—*Production of Spelter*, 1912

Source	Short Tons Spelter	Zinc "Ore"			Zinc-Lead Ore			All Other Ores		Total	
		Crude Ore	Recoverable Zinc Content	Per Cent.	Crude Ore	Recoverable Zinc Content	Per Cent.	Crude Ore	Recoverable Zinc Content	Crude Ore	Recoverable Zinc Content
Arizona	4,092	22,321	3,828	17.2	4,190	551	13.2	.....	.....	26,511	4,379
Arkansas	604	32,755	748	2.3	.....	.....	.....	.....	.....	32,755	748
California	1,672	6,472	2,160	33.4	.....	.....	.....	.....	.....	6,472	2,160
Colorado	60,841	177,946	42,495	23.9	229,107	21,339	9.3	2,169,573	2,277	2,576,626	66,111
Idaho	6,800	1,236	309	25.0	480,126	6,644	1.4	.....	.....	481,362	6,953
Illinois	3,952	147,859	4,065	2.6	.....	.....	.....	.....	.....	147,859	4,065
Kansas	5,668	744,254	10,633	1.4	.....	.....	.....	.....	.....	744,254	10,633
Kentucky	394	12,000	491	4.1	.....	.....	.....	.....	.....	12,000	491
Missouri	149,557	9,321,145	136,551	1.5	.....	.....	.....	.....	.....	9,321,145	136,551
Montana	14,196	136,552	13,431	9.8	91	28	30.8	.....	.....	136,643	13,459
Nevada	6,132	7,394	2,108	28.5	20,054	4,553	22.7	.....	.....	27,448	22,162
New Jersey	16,941	459,585	69,755	15.2	.....	.....	.....	.....	.....	459,585	69,755
New Mexico	6,882	16,394	4,012	24.5	15,792	2,771	17.5	.....	.....	32,186	6,783
Oklahoma	2,041	293,500	5,769	2.0	.....	.....	.....	.....	.....	293,500	5,769
Tennessee	1,935	32,347	2,191	6.8	.....	.....	.....	.....	.....	32,347	2,191
Texas	245	351	119	33.9	.....	.....	.....	.....	.....	351	119
Utah	7,756	9,422	2,800	29.7	216,764	5,729	2.2	.....	.....	226,186	8,529
Virginia	62	2,132	236	11.1	2,803	13	0.5	.....	.....	4,935	249
Wisconsin	34,137	1,363,636	33,050	2.4	.....	.....	.....	.....	.....	1,363,636	33,050
	323,907	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

in those mines where zinc is the only product, reaches the form of spelter. Such being the case, the question naturally arises, why do we have this loss, and is there any one branch of the industry which is responsible for more than its share of this loss? It is for this reason that we have brought up this subject at this time, and in a brief manner we have endeavored to show where and why these losses occur and why it is that so much attention, and properly so, is at present being given to the matter of developing new processes which will be suited to the treatment of zinc ores.

The Present Sources of Zinc

The zinc-reduction plants of the United States produced, from domestic ores, a total of 323,907 short tons of spelter in 1912. The ores originated in 19 States, and in tonnage ranged from 62 tons in Virginia to 149,557 tons in Missouri.

Table I shows the production of primary spelter in short tons, also the source of mine production and kinds of ore, in short tons, in 1912.

All these 19 States produced zinc ore and 8 States both zinc ore and lead-zinc ore.

The variety of these ores is almost limitless. They vary in zinc content from 33.9 per cent. in Texas to 1.5 per cent. in Missouri; from a clean-cut sphalerite in a limestone-chert gangue free from any appreciable quantity of iron-lead sulphides to an admixture of zinc carbonate and silicate, lead, copper, and iron minerals.

Table II, compiled from statistics for 1911, shows, for each Western zinc-producing State, wherever possible, the number of zinc-producing mines, the tons of crude ore produced, and the output of gold, silver, copper, lead, and zinc.

The zinc-producing ores of the United States are of many kinds. Of the total mine production of zinc in 1912, a quantity corresponding to 88.4 per cent. was derived from zinc ore, 11 per cent. was derived from lead-zinc ore, and 0.6 per cent. was recovered from all other ores.

Zinc Ores

Excepting Montana and Arizona zinc ores, nearly all Western zinc ores produced are shipped direct to the zinc smelters. During 1912 some ore was shipped from Leadville carrying as low as 20 per cent. zinc. The zinc ore shipped from other sections of the West carried well above 25 per cent. zinc. As a general proposition there is but a small profit to the Western shipper of zinc ores which contain less than 30 per cent. zinc. Sulphide and carbonate zinc ores are shipped direct to the zinc smelters, the carbonate ores forming the larger portion. The zinc ores of Butte average approximately 21.7 per cent. zinc[3] with a saving of 80 per cent.

[3] Annual Report, Butte & Superior Co., for the year ending Dec. 31, 1912.

TABLE II.—*Zinc-Producing Mines in Western States*, 1911

	Producers			Tons of Ore Sold or Treated	Mine Production of Metals				
	Zinc Ore	Lead-Zinc Ore	Siliceous Ores		Gold, Ounces	Silver, Ounces	Copper, Pounds	Lead, Pounds	Zinc, Pounds
Arizona	4			16,106	1,119.01	40,061	1,676	211,046	3,593,974
Arizona		4		10,756	1,633.33	44,783	19,329	2,223,351	960,097
Arizona			170	243,900	104,011.72	414,981	730,140	8,913	8,913
California	2			4,625					2,807,035
Colorado	no record			110,845	164.54	55,969	11,883	407,007	51,388,636
Colorado				214,385	10,215.35	476,049	160,835	23,687,597	38,224,372
Colorado				1,874,103	866,252.76	5,096,026	5,614,076	18,724,095	4,994,219
Idaho	2			51,050	7.74	33,669	6,192	1,027,140	6,622,185
Idaho		7		*397,354	446.01	510,667	111,408	31,098,194	1,718,064
Montana	2			225,586	817.15	491,161	1,340,427	846,680	43,734,150
Montana		1		128				17,446	75,995
Nevada	8			3,013	0.29	9,216	764	180,910	1,999,898
Nevada		3		4,654	14.89	32,870		1,151,750	1,598,134
New Mexico				8,402					2,204,029
New Mexico				16,333	14.00	35,870	66,898	1,557,093	8,033,147
Utah	6			5,753	1.60	491	478	9,601	1,165,064
Utah		14		280,817	4,959.31	1,358,311	1,115,770	35,100,763	16,675,197
Washington		1		51	9.00	1,602	411	11,982	20,590

*0.2151 per cent. zinc.

made in concentration. The Franklin Furnace region of New Jersey is the principal zinc-producing region of the eastern part of United States. This ore is an intimate mixture of three zinc minerals, franklinite, zincite, and willemite. Virginia and Tennessee produce zinc carbonate and sulphide. The zinc recovered from the ores of the Eastern States ranges from 108.4 lb. in Virginia to 303.5 lb. in New Jersey.

For the most part the zinc ore treated in the Central States consists of sulphide ore. The metal recovered from each ton of ore in the concentrates ranges from 38 lb. in Kansas to 72 lb. in northern Illinois.

Lead-Zinc Ores

A variety of ores is included under this heading. They range from "dry" ores high in silver with low lead and zinc values and ores with gradually increasing lead and zinc values to the complex micro-crystalline ores found in a number of camps throughout the West, a good example being those of Rico, Colo. There are some ores in the West of this nature that are shipped either to lead or zinc smelters, depending on the gold and silver values. If the percentage of zinc is sufficiently high, the shipper sacrifices the gold and silver values, but on the other hand, if the gold and silver values are high the ore is sent to a lead or copper smelter, and the shipper is not only penalized for the high zinc content of the ore, but the zinc itself is lost.

About 11 per cent. of the zinc production of the United States is recovered from lead-zinc ore. Except for small amounts produced in the Eastern States all of this 11 per cent. comes from the Western States, Colorado and Utah producing the major portion.

The only zinc-bearing siliceous or dry ore produced which is worthy of note is in Colorado. The zinc from these ores is saved in concentrating mills as a by-product.

Fig. 1 shows graphically the present source of zinc and the ores which are not being utilized.

Loss of Zinc at the Mine

The losses of zinc at the mine, in the aggregate, amount to a large tonnage. Ultimately some of these losses can be recovered. In the average Western lead mine zinc generally occurs as a shell in the stopes from which oxidized ores have been extracted. In most mines the lateral extent of these shells is unknown. To fill these stopes with waste will, in most cases, make the shells an absolute loss. As mining progresses the walls of all stopes should be sampled at regular intervals and a record kept of the results. What is too lean to treat to-day may net a profit in the near future.

In the indiscriminate dumping of low-grade ore and waste on the surface, much zincky material is lost. In the development of many lead-silver mines, considerable tonnages of zinc-iron or zinc-iron-lead

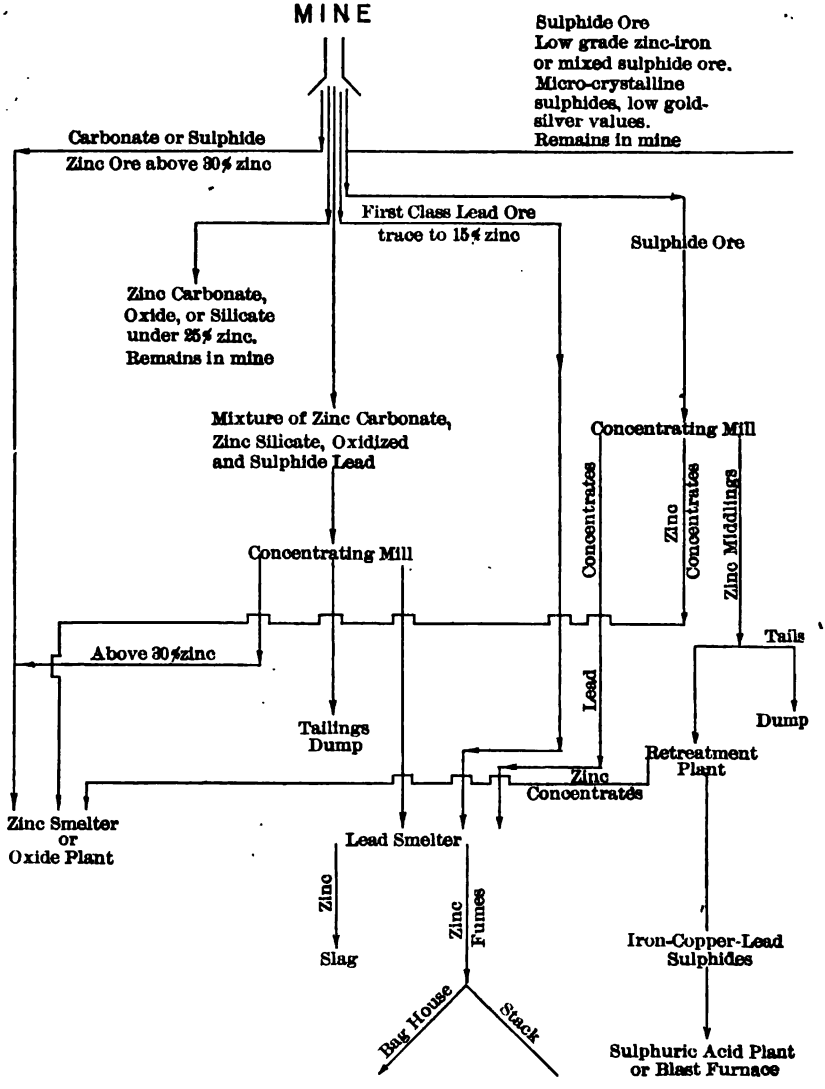

FIG. 1.—PRESENT SOURCE OF ZINC, AND ORES NOT UTILIZED.

materials, which are micro-crystalline in their nature, are often encountered in the mineralized zone adjoining the ore or mixed throughout same. Much oxidized zinc ore will be mined above the water level in many silver-lead or lead-zinc properties and in mines producing siliceous ores.

There is a vast tonnage of zinc tied up in the low-grade zinc ores showing values in excess of $5. Of this value the major portion is in oxidized zinc.

The Present Method of Extracting Zinc from its Ores

As previously stated, the retort process is the only process in commercial use at the present time which seems to meet the requirements of the chemistry of zinc; namely, the requirements imposed by the fact that the metal is reduced from the oxide at a higher temperature than is required to volatilize the metal, and for that reason it is reduced as a vapor, which must then be condensed to liquid form. Then, too, the condensation temperature must be regulated within very narrow limits, or a large part of the zinc vapor will condense as very finely divided zinc mixed with oxide (blue powder). Due to these requirements, that is, the proper regulation of the temperature of the reduction and the condensation, the retorts in which the process is conducted must of necessity be small, and this fact in turn requires the use of small charges, and for this reason, as is well known, the zinc smelters can only treat at a profit ores which contain a comparatively high percentage of zinc.

Class of Ores suited to Zinc Retort Smelting

At the present time the lower limit of zinc in ores or mill products suited to this process is about 25 per cent. Therefore the ores or products suited to the retort process of zinc smelting may be obtained from:

(*a*) Straight zinc ores;

(*b*) The concentration and separation of zinc-lead ores;

(*c*) The concentration and separation of mixed and complex ores, containing gold, silver, lead, copper, zinc, etc.

Concentration Processes Used for Treating Zinc Ores

As the ores smelted by zinc smelters are for the most part products of concentration processes, we will first consider a few of these which are in use at the present time, in order that we may have a better understanding of the problem which presents itself. For the sake of convenience, we will consider these processes in the following order:

1. Hydro-mechanical processes;
2. Electro-mechanical processes;
   (*a*) electromagnetic,
   (*b*) electrostatic.
3. Flotation processes.

Hydro-Mechanical Processes

As these processes are well known, and as the various types of concentrating machines in use at the present time have been fully described in the literature on the subject, it is not necessary to more than state that

such machines are generally used, so far as zinc ores are concerned, for obtaining a concentrated product which may afterward be separated into a zinc product and a lead product, or a zinc product and a product which may contain lead, copper, gold and silver, and which is sent either to a lead or a copper blast furnace.

From *Technical Paper No.* 41, *U. S. Bureau of Mines*, by Clarence A. Wright, we have the following interesting data showing the efficiency of hydro-mechanical concentration processes in Joplin, Mo. These ores require the simplest form of hydro-mechanical concentration.

Quantity of ore through mill, tons	309.70
Quantity of zinc concentrates produced, tons	9.235
Assay of zinc concentrates, per cent. Zn	59
Quantity of blende (ZnS, 67 per cent. Zn). in concentrate, tons	8.132
Quantity of tailings, tons	300.465
Assay of tailings from mill (0.96 per cent. Zn.), per cent. of ZnS	1.43
Loss of blende (ZnS) in tailings, tons	4.296
Total quantity of blende (ZnS) in ore through mill, tons	12.428
Total recovery of blende through mill, per cent	65.43
The average loss, therefore, in the concentration of zinc ores in the Joplin district is 35 to 40 per cent. There are heavy losses in smelting. Total loss reaches nearly	50 per cent.

Electro-Mechanical Processes

(*a*) *Magnetic Separators.*—Although there are many types of separators of this class, a magnetic separator in general consists of a set of magnets so arranged that when paramagnetic or weakly magnetic minerals are brought into the magnetic field they are influenced in such a manner as to effect the desired separation. For example, if a mixed sulphide be crushed to the required degree of fineness and then passed over a wet-concentrating table which will separate it into a galena-pyrite product and a pyrite-blende product and the latter product then be passed over a magnetic concentrator, such as the Wetherill, two products are obtained. The first, a blende product, assaying about 50 per cent. zinc, 10 to 12 per cent. iron, and 1 per cent. lead, is sold to the zinc smelters; the second, a pyrite product, containing some lead and about 5 to 7 per cent. zinc, which is mixed with the galena-pyrite heads from the Wilfley tables, is sold to the lead smelters, for whom it is a desirable ore. In smelting this ore in the lead blast furnace, the zinc is lost in the slag and fume.

(*b*) *Electrostatic Separators.*—Separators of this class are based upon the fact that while most gangue minerals and certain of the sulphides, such as blende, are relatively poor conductors of electricity, many metallic sulphides and other minerals are good conductors, and likewise upon the fact that the bodies, charged alike electrically, repel one another, while if charged oppositely they attract each other. Thus, as pointed out by Richards:[4]

[4] Richards, Robert H.: *Ore Dressing*, vol. iii, p. 1549 (1909).

"If a mixture of good and poor conductors, in a neutral state, be dropped upon a highly charged conducting surface, the good conductors immediately receive a charge similar to that of the surface and are repelled, while the poor conductors are much more loath to receive the charge and therefore not so readily repelled. If, however, a material charged to a high potential of the opposite sign be fed to the above-mentioned surface, the good conductors, as before, assume immediately the condition of the charged surface and are repelled, while the very poor conductors, carrying a charge opposite to that which the surface carries, by the above law, tend to cling to that surface, thus making a sharper division of the separation. Theoretically the separation of two minerals does not require that one of them be a very good conductor and the other a very poor conductor, but merely that there be a difference in the degree to which they will conduct electricity. Commercially, of course, there are limits to this."

An example of the working of such a separator is the work done at Midvale, Utah, near Salt Lake City, at the plant of the United States Smelting, Refining & Mining Co. This plant has been in operation since 1909, and it is stated[5] that from 5,000,000 to 7,000,000 lb. of zinc per year is saved which heretofore had gone to waste. At this plant zinc-iron middlings, partly furnished by the company's concentrator, partly from custom ore from the district, are passed over a Huff separator. MacGregor states that the "feed" to the mill assays approximately 0.04 oz. gold, 2.5 oz. silver, 2.5 per cent. lead, 1 per cent. copper, 23 per cent. iron, 28 per cent. zinc, and the separated zinc product assays 48 to 50 per cent. zinc with 3 to 5 per cent. iron, and the iron product 10 per cent. zinc. Most of the gold, silver, copper and lead is found in the iron product, and this product is smelted in the company's lead furnaces.

In this connection it is to be noted that the iron product contained about 10 per cent. zinc. The assays on other products given by MacGregor show the same thing; that is, that in the "iron product of the copper product," or whatever name is given to the product which goes to the blast furnace and from which the greater part of the zinc has been separated, there is still anywhere from 8 to 12 per cent. zinc, and, as previously stated, the zinc in this product is not recovered.

Flotation Processes

The flotation process has been briefly described as follows: A mixture of ore, oil, and water is agitated by a rotary stir; the valuable metalliferous part of the ore, through an affinity of the oil for the sulphides, will float when assisted by the air included by agitation, while the worthless gangue will sink.

Based on this principle, a selective flotation process known as Horwood's process has been devised. It is stated by the *Australian Mining*

[5] MacGregor, Frank S.: Progress in Electrostatic Ore Dressing, *Transactions of the American Electrochemical Society*, vol. xxiv, p. 352 (1913).

Standard of Jan. 15, 1914, that the successful establishment of this process at the works of the Zinc Corporation was one of the metallurgical achievements of 1913.[6]

"The principle of this process is the roasting of mixed lead-zinc concentrates obtained by flotation, whereby the lead particles are slightly sulphatized while the zinc is unaffected. On floating the roasted product the zinc alone floats and the lead remains submerged.

"A plant having a capacity of 500 tons weekly has been in operation for over six months, treating current and accumulated slime concentrates. The slimes are first dewatered and washed to remove soluble salts, then lightly roasted in an Edwards furnace, and finally treated again by flotation. Typical actual results from this plant are as follows:

	Zinc, Per Cent.	Lead, Per Cent.	Silver, Oz. per Ton
Feed to roaster	40.4	14.6	21.4
Zinc concentrate	48.7	5.2	11.6
Lead residue	10.2	44.2	54.6"

Thus we see that, as in the electrostatic and electromagnetic processes, the lead residue which goes to the blast furnace contains about 10 per cent. of zinc, and this zinc is lost.

In studying the processes above mentioned, we find that in every case the product which goes to the lead or copper blast furnace does not contain much less than 10 per cent. of zinc, as shown in Table III.

Loss of Zinc in Lead and Copper Blast-Furnace Smelting

In presenting these facts as regards the methods which are used for concentrating zinc ores, we do not wish to be understood as criticising any of the concentration processes mentioned, as they do the work for which they are installed; namely, to separate gold-silver-lead-copper-zinc bearing ores into two products, one of which shall contain enough zinc to enable it to be profitably treated by the zinc smelter, and the other, no more than will permit of its being satisfactorily treated in a copper or lead blast furnace. As is well known, in such furnaces, if the zinc content of the ore constituting the charge is not greater than 10 per cent., it can be gotten rid of in the slag and fumes. The loss of this amount of zinc is, of course, a distinct waste. As to the amount of zinc lost in this manner, the following example will serve to illustrate the same.

If we treat ore containing, say, 28 per cent. of zinc, we separate this into two products, one of which will contain, say, 50 per cent. zinc, while the other will contain most of the gold, silver, copper, and lead, and about 10 per cent. of zinc. In other words, of our original product of 50 tons we will have 28 tons (more or less, dependent upon the per cent. of iron pres-

[6] *Metallurgical and Chemical Engineering*, vol. xii, No. 5, p. 350 (May, 1914).

Table III.—Recovery of Zinc by Various Concentration Processes

Process	Per Cent. Zinc in Zinc Concentrate	Per Cent. Zinc in Product going to Blast Furnace	Per Cent. Recovered			
			Zn	Pb	Ag	Rem.
Ordinary Wet Concentration:						
Daly West, Park City, Utah.			50.0	95.0	75–80	
Daly Judge, Park City, Utah.			66.0	92.0	78.0	
Buffalo Hump Mining Co. Burke, Idaho			25.0	90.0	85.0	
Joplin, Mo			65.43			
[a]Wetherill—Magnetic	50.0	5–7				
[b]Huff—Electrostatic	48–50	10.0				
	41.3	12.6				
	49.3	8.6				
[c]Potter-Delprat			66.2	38.5	50.9	
[d]De Bavay 1910	48.1	11.7				
[d]De Bavay 1911	48.4	14.2				
[d]De Bavay 1912	49.2	16.6				
[e]Elmore Vacuum	46.0	15.84				
Mineral Sep. Process			85.4	74.8	80.1	
Horwood Process	48.7	10.2				
Hyde Process at Butte and Superior			88.1 86.4	..... 62.5	83.4 80.0	For 1913

[a] Magnetic Separation of Zinc Blende at Denver, Colo. *Engineering and Mining Journal*, vol. lxxiv, No. 7, p. 217 (Aug. 16, 1902).

[b] MacGregor, Frank S.: Progress in Electrostatic Ore Dressing. *Transactions of the American Electrochemical Society*, vol. xxiv, p. 352 (1913).

[c] Hoover, T. J.: *Concentrating Ores by Flotation*, p. 91 (London, 1912).

[d] *Idem*, p. 97.

[e] *Idem*, p. 105.

ent) of product containing 50 per cent. zinc, and 22 tons containing 10 per cent., and in addition most of the gold, silver, copper, and lead contained in the original product. This latter product goes to the blast furnace and ordinarily the zinc content is not saved, but is lost in the fume and slag. This amount of zinc therefore represents a loss of almost 1,500,000 lb. a year. Moreover, this zinc is lost for all time, as it cannot be recovered, and when we consider that this is lost from the treatment of only 50 tons of concentrates per day, we can readily understand, as has been shown by those who have investigated this subject, that the amount of zinc lost in this way amounts in the aggregate to thousands of tons annually, and, as has been pointed out by Clevenger[7] and others, in the

[7] *Metallurgical and Chemical Engineering*, vol. xii, No. 5, p. 299 (May, 1914).

inability to recover this zinc lies one of the greatest weaknesses of our present-day metallurgical practice, in that the zinc so lost represents a distinct economic waste, much of which will never be recovered, and that we must look to the prevention of this waste as one of the sources from which we shall in the future obtain a large portion of our zinc supply.

The same thing is of course true to some extent of lead, for in copper smelting, if lead is present, it is volatilized and no attempt is made to recover it, and so the lead is completely lost. As before stated, these facts are not presented with the idea of bringing an indictment against electro-mechanical methods of concentration, or against modern methods of copper and lead smelting, but rather to emphasize the fact that although mechanical methods of concentration and modern methods of smelting have reached a very high stage of development, their use nevertheless entails a very great loss of zinc, which is not recoverable, and that therefore it is quite likely that in the future it will be necessary to stop these wastes in order to add to the available supply of zinc.

Possible Future Sources of Zinc Supply

A study of the data as given in the preceding tables, etc., brings out two points very clearly; namely, that only straight and mixed low-grade sulphide ores are at the present time being concentrated with any degree of success, at least so far as effecting a recovery of the zinc content of the ore is concerned, and, second, that low-grade carbonate ores—that is, ores containing less than 20 per cent. zinc—are not treated at all. Therefore, we may say that at present zinc ores are not being treated by any process except the retort process, and that this process cannot profitably treat ores containing less than 20 per cent. zinc. Thus we see that, so far as zinc ores themselves are concerned, we have as a possible future source of zinc:

1. The values which are now wasted as a result of imperfect concentration methods;
2. The zinc which is lost in the slag or as fume when ores containing an appreciable amount of zinc are smelted in copper and lead blast furnaces;
3. The low-grade carbonates which cannot be treated at a profit by present-day metallurgical processes;
4. The micro-crystalline sulphides of zinc, iron, and lead; *e.g.*, such ores as are found in the Commonwealth mine, near Reno, Nev.; in some of the Pioche, Nev., mines; and in Rico, Colo.

It is true that ores of the latter class containing sufficient values in lead and silver may be mined and shipped as lead-silver ore, but if so, no attempt is made to recover the zinc; in fact, in such cases it constitutes an interfering element and is lost in the slag or as fume when such ores are smelted in copper and lead blast furnaces. Moreover, the shipper is

penalized at so much per unit for the zinc content of his ore. For this reason, as above stated, we may consider the zinc that is lost in this way (that is, as slag and fume in the smelting of ore where zinc is considered a harmful impurity) as one of the possible future sources of zinc.

The Problem of the Metallurgist and the Present Status of New Processes

If these are the possible sources of zinc, the problem that presents itself is how to utilize them in a commercially feasible manner, and it is one to which is being given a great deal of time, attention, and money.

Perhaps no one process can be devised which will solve it, but that on the other hand local conditions, and the nature of the ore to be treated, will determine in each particular instance whether an igneous concentration, a chemical, an electro-chemical, an electro-thermic, or some other process shall be used which does not belong to any of the types of processes just mentioned. As above stated, a great deal of work has been done in an experimental way on all of the types of processes above mentioned, and inasmuch as they have been quite fully discussed from time to time in the technical journals and before this and other societies, it is not necessary to do so in this paper.

As to the present status of these various processes, we can perhaps do no better than to quote from an editorial on "Developments in the Metallurgy of Zinc in 1913," which appeared in the January number (1914) of *Metallurgical and Chemical Engineering:*

"From a metallurgical point of view the status of spelter production was not materially changed during 1913. This is not to imply a dearth of experiment and research in the winning of zinc; on the contrary, there was marked activity in almost all branches of zinc-ore treatment. Some hopes have been realized, and progress has been recorded; but a reduction process has not been brought forth that can be considered a serious competitor of retort smelting.

"On the whole a commendable lot of work has been done toward improving present processes and in attempts to develop new ones. A great deal of time and money have been spent in investigation, and there is apparent willingness to test new ideas of merit. Apparently no line of development has been finally dropped, and the coming year should witness a nearer approach to the realization of the hopes and ambitions of different investigators."

The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process

BY S. E. BRETHERTON, SAN FRANCISCO, CAL.

(Salt Lake Meeting, August, 1914)

MOST metallurgists appreciate the great need of a process for the extraction and recovery of valuable metals from complex ore, where the presence of one metal increases the cost of extracting the other metals to such an extent that not any of the metals can be extracted at a profit. We need only refer to some of the mines in Colorado, Utah, and New Mexico, where oxidation has taken place and the sulphur, zinc, copper, and some of the iron and other metals have been leached out, large bodies of oxidized ore remaining, rich in lead, silver, and sometimes gold. These metals have been concentrated by the removal of the other metals, leaving a very desirable ore for treatment in the blast-furnace for the production of lead bullion, and recovery of the silver and gold. If lead is not present, the gold and silver metals are usually recovered by some method of milling, or the ore is shipped to a smelter, where it is mixed with lead ore or copper-iron sulphide ore from some other mine. In these same mines, when developed to greater depth below the leached or oxidized zone, the ore is found to contain the original mixture of sulphur, zinc, copper, etc., added to, or I should say left with, the lead, silver, and gold values; but instead of adding to the value of the ore, the presence of these elements, especially zinc, makes the copper, silver, and gold of less value, and often worthless. Such was the experience of the oldest mines in Leadville, Colo., until methods of mechanical separation of the lead from the zinc were developed sufficiently to allow of some profit by the making of a concentrate rich in zinc to be shipped to the zinc smelter, and sometimes a lead concentrate which could be shipped to the lead smelter; but even in these cases the waste of the different metals is a very serious matter. In the very best concentrating plants some zinc will be left in the lead concentrate, to be lost and to give considerable trouble in the lead-smelting furnace. On the other hand, some of the lead, with a large percentage of the copper and silver, will follow the zinc into the zinc concentrate, to be, perhaps, recovered by smelting the cinder or clinker left in the re-

torts after volatilizing the zinc for the production of spelter. The losses of copper and gold in this case are not so great, but the cost of smelting this residue is often greater than the combined values of the gold, silver, and copper in it. The loss of silver by volatilization is also much greater than the loss of either gold or copper, so that zinc ore rich in silver is not suitable for this process. On the other hand, a sulphide ore containing the same amount of zinc as the above complex ore, with none of the other metals, can be easily concentrated. Often it is a simple matter to make a concentrate containing from 40 to 60 per cent. zinc (as is now being done in the middle Western States) which can be sold to the zinc smelters, who will pay a much higher price per unit for the zinc contents at 60 per cent. than at 40 per cent. Any one familiar with metallurgy will appreciate that one reason for this is, that the higher the percentage of a metal in an ore, the higher will be the percentage of recovery obtained.

A sulphide copper ore containing, say, 3 per cent. and upward of copper and little or no zinc is very cheaply reduced to a copper matte, if in sufficient quantity and within a reasonable distance of some railroad. The presence of silver and gold is of great benefit in assisting the copper to make profits, for the reason that the freight on precious metals, when shipped with either copper or lead products, is almost negligible. But supposing such ore to contain 14 to 40 per cent. zinc with barium and other impurities, and the structure of the ore not suitable for any mechanical means of concentration; in such cases the zinc content decreases, and sometimes destroys, the value of the copper, while the ore is not rich enough in zinc for the recovery of zinc by any ordinary methods. Such ore as this, containing copper, lead, silver, gold, and zinc, occurs in a number of large mines in California, Colorado, Utah, Nevada, Arizona, and New Mexico, some of which mines are being worked successfully by concentrating the ore into zinc concentrates and lead concentrates, with sometimes a copper concentrate; but the per cent. of loss of the different metals is excessive. There are a great many mines the ore of which cannot be concentrated by any known method. I have had such ore as this in mind for several years, and after investigations I decided to have the ammonia-carbon dioxide process tested and developed. This process is the only one I know of for the treatment of the complex zinc sulphide ores of the Rocky mountains by which the gold, silver, copper, and lead can be recovered and saved, after first extracting and saving the zinc, thus turning the zinc into an asset instead of a liability, and enabling a great number of mines now idle to become profitable.

There is another view to be taken of this situation: The United States government should and perhaps will insist on all railroad companies and large mining companies treating their timber to prevent decay, except, perhaps, that used for temporary structures; first, to lessen the number of accidents, and second, to stop the unnecessary waste and destruction

of timber in the country. This would not be any hardship on either the railroad companies or the mining companies, for, looking at the subject from an economic standpoint, it would be a great benefit to them to have their timbers last from 12 to 20 years, instead of only 2 to 5 years, as is often the case at present. The high price of creosote almost prohibits its use, except perhaps as an outside coating for timber already treated with a cheaper wood preservative, such as zinc chloride. (The question is, could not some cheaper oily substance be used for a surface coating to timbers in wet ground to prevent the leaching out of the zinc chloride?) I have mentioned the use of zinc chloride for wood preservative purposes to show that there is a demand for zinc in this direction, which will help to solve the problem of marketing the zinc when recovered from the immense quantities of complex zinc ore in the Rocky mountains. At the same time that the zinc is being recovered from such ore, considerable quantities of lead, copper, silver and gold will be added to the world's supply. In addition to the demand for zinc to be converted into zinc chloride for wood preservative purposes, the use of the metal for roofing, for zinc foil for wrapping tea and other packages, and for the manufacture of brass is constantly growing; as is also the use of zinc oxide for the manufacture of pigments for painters, and in the manufacture of rubber goods.

W. R. Ingalls, in his *Metallurgy of Zinc and Cadmium* (p. 668) writes:

> "The merits of zinc white as a pigment were summarized by Maximilian Toch in a paper recently read before the New York Section of the Society of Chemical Industry. He stated that the consumption of white lead is decreasing and that it should not be used alone. The specifications of the United States Lighthouse Department demand a mixture of 75 per cent. of zinc oxide and 25 per cent. of white lead. Zinc white mixes well with other pigments, particularly with white lead. Zinc oxide made directly from ore is more durable than that made by burning spelter."

The process[1] I have in mind for the treatment of the complex zinc ores is briefly as follows: Grind the ore, roast, regrind, leach out zinc with ammonia and carbon dioxide, precipitate from the solution any copper which has been dissolved with the zinc, and put back in the residue. Smelt the residue for the recovery of copper or lead, with the gold and silver. The ammonia and excess carbon dioxide are distilled from the solution containing the zinc and recovered for further use. As the ammonia is being distilled, the zinc is precipitated as a pure white basic zinc carbonate, filtered out, and then calcined to zinc oxide, or the basic zinc carbonate can be reduced direct to either zinc chloride or spelter. Due to lack of funds, we have not yet installed a large commercial plant at the Afterthought mines, in California, so that at present we only have a 150-lb. capacity plant.

Since writing my last article for the Institute, our metallurgical chem-

[1] *Trans.*, xlvii, 82 (1913).

ist, F. L. Wilson, has made a number of continuous runs, each of several days' duration. The following is a copy of his last report of a 14 days' continuous run: (It should be noted that this report is based on a test of our lower-grade zinc ore containing about 20 per cent. zinc. The extraction from our ore containing 30 to 36 per cent. zinc is from 85 to 95 per cent.)

Ingot, Cal.,
March 6, 1914.

S. E. Bretherton,
Mills Building,
San Francisco, Cal.

Dear Sir:

I herewith report results on test No. 65 secured from 12 complete cycles and during a period of 14 days. Two days were required for unavoidable repairs.

Preparation of Ore

The raw sulphide ore was crushed to 30 mesh (maximum). Its sulphur (soluble) content varied from 18.7 to 20.5 per cent. and averaged 19.7 per cent.

A sulphatizing roast was given the ore. The weight of the roasted ore showed a slight increase over the raw ore in several cases, the average weight remaining nearly the same.

The roasted ore was wet slimed for 10 hr. in each case. No attempt to classify the product was made. The coarse product remaining on 150 mesh was but little in each case.

Extraction

Pressure throughout agitation, filtering, leaching, etc., was 30 lb.

Time throughout the cycles was less than 8 hr. average. This time was inclusive of time taken to operate valves, etc.

Time of Cycle

	Agitation	Filtering	Leach Wash	Water Wash	Total Time
	Hr. Min.	Min.	Hr. Min.	Hr. Min.	Hr. Min.
A.....	4 20	50	1	... 45	6 55
B.....	3 15	20	1	1	5 35
C.....	3	13	2	2	7 13
D.....	3	20	3 30	1 30	8 20
E.....	3	18	3 40	... 50	7 48
F.....	3	20	4	... 20	7 40
G.....	3 05	20	3 30	... 15	7 10
H.....	3 15	20	3 25	... 20	7 20
I.....	3	20	4	... 20	7 40
J.....	4 22	25	3 30	... 30	8 47
K.....	3	25	3 40	... 30	7 35
L.....	3 35	35	40	... 30	7 20

Solution used in agitation was about 4 : 1 ore, due to the small charges used. On a 19 per cent. zinc ore the filter press in use will handle conveniently the residue of a 30-lb. charge; whereas, with a 25 per cent. zinc ore a 40-lb. charge, and 30 per cent. zinc ore a 50-lb. charge. This is due to the relative shrinkage in volume of the residue when the zinc is extracted.

Zinc extraction varies from 79.7 per cent. on an 18.6 per cent. roasted zinc ore to 84.9 per cent. on a 20.72 per cent. roasted zinc ore.

Copper extraction was not recorded, as the cement copper was filtered with the residue.

Silver extraction is negligible, which remains in the residue. The slight extraction is no doubt due to discrepancies in the weights of the net ore and residue.

Analysis and Extraction

	Roasted Ore				Residue				Extraction	
	Ag Oz.	Cu Per Cent.	Zn Per Cent.	S Per Cent.	Ag Oz.	Zn Per Cent.	S Per Cent.	Loss in wt. Per Cent.	Ag Per Cent.	Zn Per Cent.
A	7.56	2.76	19.00	5.92	12.34	8.20	0.35	39.1	0.53	73.7
B	8.28	2.96	19.40	6.33	13.20	7.60	0.41	33.8	Nil	74.4
C	8.16	2.84	20.30	6.25	12.76	7.40	0.39	42.4	0.98	80.8
D	8.16	2.84	20.30	6.25	12.70	7.20	0.41	37.3	0.24	78.1
E	8.20	3.12	19.80	5.04	12.96	6.90	0.55	42.9	0.98	80.2
F	7.48	2.96	21.40	6.84	11.20	6.52	0.58	41.4	1.21	82.1
G		3.00	20.60	6.74		6.75	0.66	42.9		81.0
H			20.80	6.78		6.82	0.62	43.8		81.6
I			19.7	7.56		6.55	0.67	41.2		80.6
J			18.60	7.02		6.86	0.62	45.2		79.9
K			19.30	7.30		6.45	1.10	44.0		81.3
L			20.72	7.74		6.45	0.77	51.1		84.9

Recovery

Boiling out with the use of the extra still kept the amount of solution down to a minimum. Live steam was admitted into the third still at intervals to facilitate the boiling out of the zinc. The excess solution was used in washing the residue and saved to grind the ore in.

Zinc precipitate was off color for the most part. The steam from the first still carried over rust and oil enough to discolor. The rust was caused by the filter press and pipes having been idle for some months previous to this test. Also fine rust and ore were carried into the receiving still from the ore filter. Decantation from the copper precipitator was tried, but the rust occurred as before. By refiltering the solution through a finer filter (filter paper placed on the leaves of the press) the resultant zinc carbonate was as white as could be desired. The volatile moisture, etc., contained in the precipitate varied from 65.2 per cent. to 78.6 per cent. The recovery of the zinc basic carbonate and hydrate was 99.7 per cent. This 0.3 per cent. was left in the solution.

Zinc Recovery

	Zinc Precipitate			
	Weight	Zinc	Zinc Recovered	Zinc Extracted
	Pounds	Per Cent.	Pounds	Pounds
A	9.05	53.2	4.82	5.94
B	7.86	53.0	4.17	3.68
C	12.23	54.4	6.66	3.86
D	5.24	53.6	2.83	4.00
E	9.20	53.8	4.95	4.58
F	12.65	53.0	6.71	5.34
G	4.94	56.0	2.76	4.44
H	9.71	51.0	4.96	5.78
I	12.70	52.2	6.64	5.59
J	6.94	51.6	3.58	4.48
K	4.56	53.2	2.43	4.49
L	13.90	55.4	7.70	6.32
		Total	58.21	58.4

Losses

Ammonia losses were 4.44 per cent. during the run (not counting A, B, C and D). Of this amount 0.28 per cent. was unaccounted for numerically, but due to intermittent leaks in the system, 17 per cent. of the total ammonia was tied up as fixed ammonia. No lime was used to free this in the run. Compressed air was used to drive the charge through the filters and the air remaining in the agitator and copper precipitator was released through a scrubber. In like manner the absorbers and separator were connected with scrubbers. Washing the residue first by hot solution from the first still and then with steam brought the residual ammonia down to from 1.3 lb. to 3 lb. per ton of ore. The zinc precipitates held some back as a salt, there being no smell of volatile ammonia.

	Per Cent.
Caught in scrubbers (lutes)	2.25
[a]Remaining in residue	0.67
[a]Remaining in zinc precipitate	1.24
[a]Unaccounted for	0.28

Carbon dioxide losses were mostly in the residues as tabulated below. The total loss was 24.72.

[a] This loss of ammonia, with the exception of the trace lost in the residue and precipitate, was higher than usual with us, and was due to defective old valves and pipes, and the use of compressed air. Practically all of the ammonia can be saved by drying the residue and zinc precipitates. The carbon dioxide can be recovered by calcining products.

	Ammonia			Carbon Dioxide	
	Residue Per Cent.	Pounds Per Ton of Roasted Ore	Zinc Precipitate Per Cent.	Residue Per Cent.	Zinc Precipitate Per Cent.
A.....	.....	.....	1.43	6.68	4.8
B.....	.....	.....	1.99	7.00	4.8
C.....	.....	.....	2.54	7.70	3.8
D.....	.....	.....	1.5	6.00	2.2
E.....	0.525	10.5	0.62	11.00	5.8
F.....	0.430	8.6	0.80	8.60	2.4
G.....	0.355	7.1	0.50	8.20	3.4
H.....	0.150	3.0	1.22	7.40	4.4
I.....	0.140	2.8	0.80	11.40	3.2
J.....	0.125	1.5	1.00	8.60	2.0
K.....	0.100	2.0	1.20	9.20	3.0
L.....	0.065	1.3	0.70	7.60	6.2

Remarks.—In recovery of the ammonia by boiling out, the ammonia content of the liquor in the absorbers could be controlled at will; 15.3 per cent. ammonia was obtained in one instance in solution to be used again.

Respectfully submitted,
F. L. Wilson,
Metallurgical Chemist

Electrostatic Separation at Midvale

BY H. A. WENTWORTH, BOSTON, MASS.

(Salt Lake Meeting, August, 1914.)

THE Huff electrostatic plant of the United States Smelting Co., operated in conjunction with its wet concentrator at Midvale, Utah, was the second plant of substantial size installed using the Huff process, and the first plant to be put in operation on the so-called Western complex ores.

In spite of the machinery being of early design and consequently embracing many of the mechanical weaknesses incident to a pioneer plant the mill has operated steadily and uniformly since the summer of 1909, about five years now, without any material change other than some rearrangement of the machinery for better handling of the sizes.

The ore, at present coming almost entirely from the company's mines in Bingham, is of practically the same composition as that upon which the plant was first put in operation.

The crude ore, consisting of the sulphides of copper, lead, zinc, and iron, and containing small amounts of gold and silver, is brought by train and delivered to the hoppers of the wet concentrator, where by jig and table treatment, there are produced a shipping lead concentrate, a tailing, and a middling product, the latter being conveyed in push cars to the adjoining electrostatic mill.

The crude ore delivered to the wet concentrator contains from 6 to 9.5 per cent. zinc. At times the plant is used also for the treatment of custom ores from the Bingham district and elsewhere, and at such times the results of the plant vary according to the material in use, but as by far the larger portion of the product is that from the company's own mines, the results given below are fairly indicative of the general work obtained.

The diagram, Fig. 1, illustrates graphically the flow of the ore through the electrostatic mill. The middling coming from the wet mill, containing from 15 to 18 per cent. moisture, is hoisted while on the cars by an elevator, and delivered from the top of the elevator to a hopper over the drier. The drier first installed was of too low capacity to take care of the moisture present in the tonnage to which the mill was later

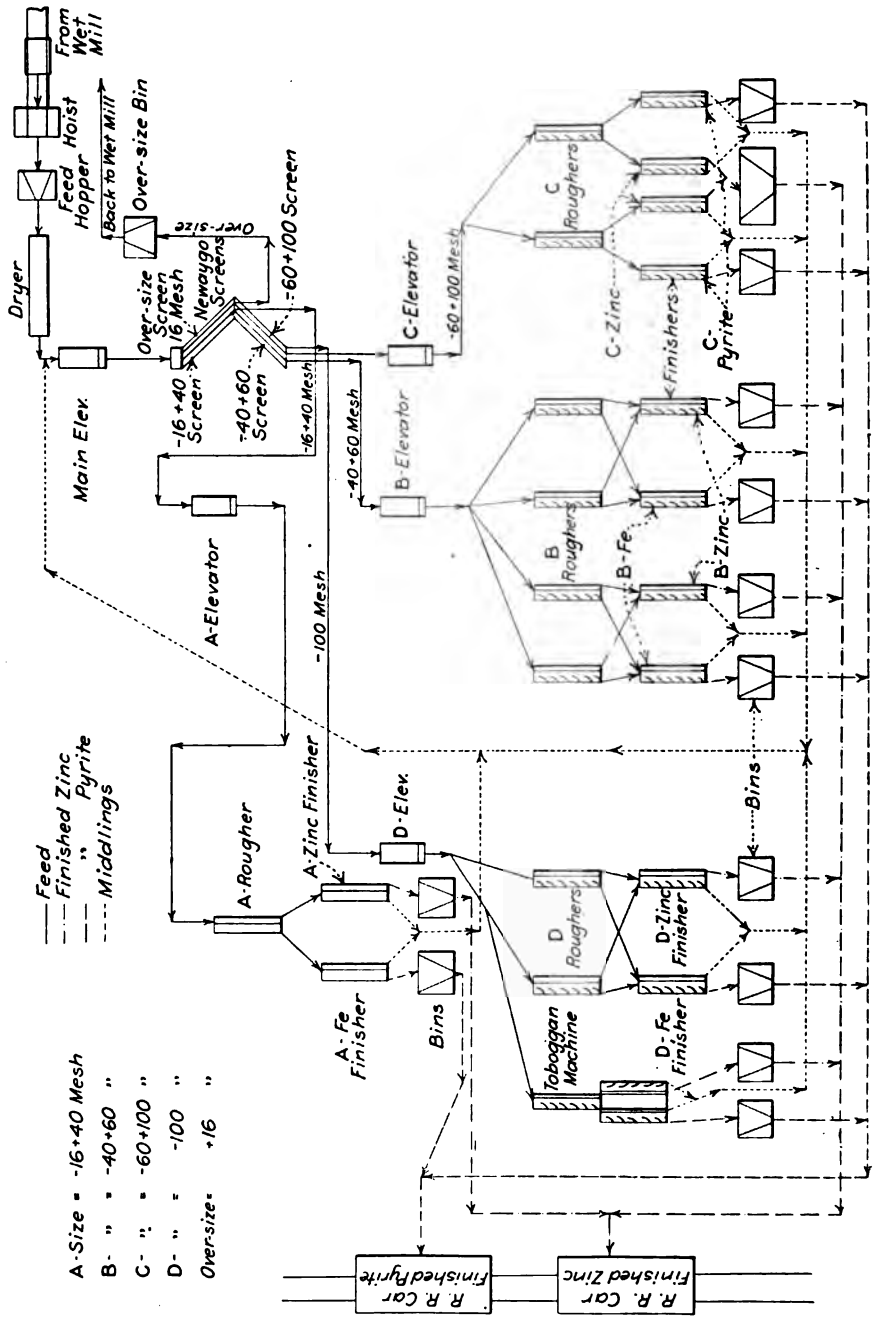

FIG. 1.—FLOW SHEET OF ELECTROSTATIC MILL.

raised, and a second drier was placed in series with it. Drying material of this nature requires in the neighborhood of 1 lb. of coal for 5 to 7 lb. of water to be evaporated; the higher the moisture the more efficient the use of the fuel, due to the heat wasted in raising the temperature of the ore as a whole, being a constant.

The drier delivers the bone-dry material to the boot of the main elevator, where it is raised, and delivered at the top of the mill to two Newaygo screens set in series, each having a double vibrating surface. The feed to these screens is practically all through 16 mesh, and mostly through 30 mesh. The top vibrating screen of the first Newaygo is a 16-mesh scalper, removing the oversize material, chips, etc., which are delivered to a bin and returned again to the wet mill. The Newaygos produce four sizes, through 16 on 40 mesh, through 40 on 60 mesh, through 60 on 100 mesh, and through 100 mesh. Of the total feed, 16 per cent. passes a 200-mesh screen; and of the material in the through 100-mesh size, which is later treated by the so-called fines machines, 46 per cent. is through a 200-mesh screen.

From the Newaygos, which deliver on to the second machine floor of the mill, each size of material passes to its separate elevator and is delivered on the third floor to small hoppers which in turn empty into the separators below.

The separators are for the most part arranged in units of three, one being above the other two. Other arrangements are indicated on the flow sheet, the idea being to make on the first machine a rough split of the minerals to be separated, and the final cleanup on the separators below. The arrangement of the units is largely dependent on the percentage of the minerals to be separated in the given material, and the machines are arranged as nearly as possible to keep a uniform load.

Each unit produces three products, a finished "iron," a finished "zinc," and a middling product. This middling product is returned to the boot of the main elevator and again passed into the Newaygo screens. This middling is due partly to mechanical entanglement, and partly to the fact that abrasion in passing through the mill has broken up some of the minerals; so by passing all the material again over the screens, it unites with the warm material coming from the drier and is therefore again thoroughly dried and screened into its proper sizes.

The finished products fall into bins under the main floor of the mill, and are thence conveyed by hand cars to the railroad cars outside the mill building.

The separators are electrically energized by a small self-contained electric unit maintained in a dust-free room at the side of the mill. This unit consists of a 3-h.p. motor, belted to a special generator and exciter. The potential is raised by transformer and rectifying device to 18,000 to 22,000 volts, which is the potential used on the machines (at

this plant). The potential generated in this way is practically uniform, and varies only with such slight variations as are present in the speed of the motor. One terminal of the potential line is grounded direct and the machines are likewise grounded. From the other terminal, a highly insulated wire runs through the mill, being connected to the so-called "attracting" electrodes of the separators, of which there is one to each roll or separating element. The electrical equipment in this plant is considerably more massive than the equipments now being put on the market.

The principle of the process and the general apparatus in use have been described in some detail during the past four years in several publications and were given by the writer at the New York meeting of this Institute in February, 1912. Numerous improvements have been made since that time, one of the most important from an operating standpoint being the substitution of bare-metal "electrodes," which has been made possible by the institution of a regulating device in the circuit to the machines. This substitution of the metal for the wooden electrodes at Midvale has reduced the cost of maintenance of the electrodes 75 per cent. Further improvement along these lines has been made, and it is believed that shortly any cost for electrode maintenance will be entirely eliminated. Since the construction of the Midvale plant, a separator has been designed called the "Toboggan" type. The only moving part in this is the feed roll; belts and bearings being eliminated. One of these separators is at present in operation at Midvale and accomplishes about the same result as a unit of three of the roll type. Elsewhere this type is in operation handling all the material through 20 mesh.

The quality of the work in the Midvale plant is largely dependent upon the character of the blende, which varies from time to time. The zinc sulphide crystals in the ores from the United States company's mines contain chemically combined from 3 to 5.5 per cent. iron. When the iron content of the zinc sulphide crystals is under 4 per cent., it is not difficult to make a 48 to 49 per cent. blende and at the same time keep the zinc content of the pyrite product around 11 per cent. or under. When, however, the zinc sulphide crystals contain much over 4 per cent. iron, it is impossible to keep the grade of the blende up above 48 per cent. unless considerable zinc is run into the pyrite product. The zinc present in the pyrite product consists partly of attached particles and partly of an intimate mixture of zinc-lead mineral, and partly the blende high in iron. The impurities left in the zinc product are largely gangue, left in the middling in the wet mill, and small amounts of attached minerals. That the percentage of iron chemically combined with the blende is not a criterion in judging the conductivity of the blende, is illustrated by the fact that some blendes, containing as high as 14 per cent. iron in

the crystals, are separated electrostatically. It is not known definitely what determines this conductivity.

Typical analyses of the wet-mill feed, and the head and resulting products of the static mill are as follows (the impurities of minor metals in each of the products being largely as minute attached particles):

	Au	Ag	Pb	Cu	Fe	Zn	SiO_2
Head, wet mill	0.08	3.8	8.4	0.41	14.3	9.0	28.8
Middling to static	0.07	3.4	2.2	0.96	21.6	26.8	4.6
Blende	0.02	2.0	1.1	0.71	5.4	47.8	8.3
Pyrite	0.10	3.8	2.6	1.63	32.0	12.0	2.4

The capacity of the present mill when running full is approximately 65 tons, and on the various sizes the units have the following capacity: on 40 mesh, 12 tons; on 60 mesh, 10 tons; on 100 mesh, 8 tons; through 100 mesh, 5 tons. The Midvale ore is a comparatively difficult one with which to obtain the best results, and the tonnage per unit is therefore lower than with a more simple ore. The separators for crude ore are of much larger capacity.

Six mills of this general type are in operation in the Western zinc field, some modifications being required to meet local conditions in each case; as well as plants in operation on other classes of work as described in the articles above mentioned.

The writer is much indebted to Messrs. Heintz, Anderson, and Lemke for their kind assistance in furnishing recent data on the plant, and for preparing the flow sheet of the mill.

Separation of Lead, Zinc, and Antimony Oxides

BY RICHARD D. DIVINE, SOUTH CHICAGO, ILL.

(Salt Lake Meeting, August, 1914)

IN the Parkes process of extracting precious metals from lead, zinc is added to the molten lead containing gold, silver, copper, and some antimony. These metals, with the exception of antimony, form an alloy with the zinc by reason of their greater affinity for it than for the lead, and this alloy, being lighter than lead, rises to the top, carrying the precious metals with it, whence it can be removed by skimming. These skimmings are further treated for the separation of the zinc therefrom, and the recovery of the precious metals, the zinc being driven off by heat and again used for another operation. In addition to the zinc that combines with the silver, etc., a certain amount is necessary to saturate the lead, and it is therefore necessary to treat the desilverized lead to free it from this residual zinc.

To do this it is heated again in a reverberatory furnace and air or steam is blown through it. This oxidizes the zinc together with a small amount of lead and antimony, leaving the lead free from zinc and antimony, and forming oxides of these metals, which rise to the top and are skimmed off.

This process relates to the separation of the metals in these oxides or skimmings.

The skimmings are cooled, crushed, and mixed with carbonate of soda and pulverized oil coke. The mixture is heated in a reverberatory furnace on a bath of lead, to about 1,000° C. The metals are reduced by the coke, the lead and antimony go into the lead bath, while the zinc is volatilized and burned to oxide.

This impure oxide is at the present time treated at Omaha by the Hall process of electrical precipitation, making a very pure metallic zinc. It can also be very easily refined by the old Schnabel process of using ammonia and carbon dioxide as a solvent. It seems to be in an ideal form for this purpose, and very few changes from the original method are required to obtain the zinc oxide practically C. P., leaving the lead and antimony as residues.

There has been considerable difference of opinion regarding the effect

of the carbonate of soda in this process. That it is of material assistance is shown by the following experiments: Skimmings mixed with carbon alone and heated gave an extraction of about 10 per cent. of the zinc, while extractions as high as 92 per cent. were obtained when carbonate of soda was added to the mixture. Various explanations of this action have been suggested, the most probable one being that the extreme fusibility of the soda salt causes it to act as a protective coating, permitting oxidation by the air in the furnace; it also gives sufficient mobility to the mass to allow of good contact between the carbon and oxides.

The skimmings from the refining furnaces vary somewhat in zinc contents, due to different methods of working at the various plants.

At the National plant of the American Smelting & Refining Co., the zinc will average from 12 to 14 per cent., while at Omaha it is somewhat lower. In the first experiments the crushed skimmings were treated in the furnace without a bath of lead, the reduced lead forming the bath. The analysis of this lead showed it to contain 8.70 per cent. antimony, 91.25 per cent. lead, and less than 0.05 per cent. zinc. The soda skim resulting from these operations amounted to about 15 per cent. of the original charge. The resulting zinc collected in bags gave an extraction of between 80 per cent. and 85 per cent. of the zinc contents, and the analysis showed it to contain: Zn, 60 to 65; Pb, 12 to 18; Sb, 1.00 per cent. It was a light, fluffy, yellowish product, which when first collected contained some CO_2.

The present practice of treating the skimmings is as follows: After the furnace has been skimmed, the skimmings are cooled and put through a crusher set to about $\frac{1}{2}$ in. The material is then mixed with finely crushed oil coke and sodium carbonate. The charge is made up approximately of 1 ton skim, 175 lb. soda ash, and 400 lb. coke. It is charged into a reverberatory furnace on a bath of molten lead. After the charge is started it is necessary to maintain the heat at the same point, as any cooling will stop the reaction, and to start it again requires much time and trouble. For this reason it has been found advisable to use oil as a fuel in place of coal.

It would undoubtedly be possible to use producer gas for fuel, or possibly powdered coal. The success of the process depends upon the maintenance of a steady, uniform heat. The charge is stirred or puddled at regular intervals, only one working door being opened at a time to avoid cooling the furnace unnecessarily. After the zinc has burned off, which can easily be determined by the absence of any bluish flame, the oil is shut off and the furnace immediately skimmed. The resulting "soda skim" amounts to about 10 to 15 per cent. of the original charge. The furnace is then ready to be recharged. The antimonial lead is tapped from a lead well as often as necessary and taken to a refining furnace for the removal of the antimony.

The analysis of the "soda skim" shows it to contain a somewhat higher percentage of zinc than the original skim, but we hope in time to improve on this. It was thought that it might be possible to save the expense of crushing and charging the skims, by simply adding the mixture of soda and carbon to the molten skim in the refining furnace itself, and although the percentage of extraction was good it was found that the repairs to the furnace were excessive, the lining being very quickly eaten out. The opposite is true: when the skim is charged cold the repairs to the furnace are very small.

Heretofore the zinc oxide in the skimmings has been lost, as there has been no way of separating it from the oxides of lead and antimony and saving it. The presence of zinc has also been a nuisance in clogging the smelting furnace by forming deposits on the sides, which retain other metals, fluxes, etc., and gradually make the furnace smaller and curtail the output.

At the National plant we have obtained a higher extraction of the zinc by skimming in two stages. Air is blown through the molten lead until the zinc is all oxidized and rises to the top. It is then skimmed off and kept by itself. The temperature of the furnace is then raised, air is blown through the bath again, and a second skim is removed, consisting largely of antimony with very little zinc. This is sent direct to the antimonial furnace to be treated.

The first skim requires less soda and carbon per ton treated, works better in the furnace, the zinc extraction is higher, and the soda skim is less.

We found that by making two skims we were able to raise the percentage of zinc from 12 to 14 per cent. to about 17 or 18 per cent. The antimony is oxidized after the zinc and most of it remains with the lead after the first skimming is taken off. The second, or antimonial, skim contains but a very small percentage of the original zinc contents. The loss of zinc being so small it is good practice to follow this method.

It requires about 24 hr. to work a charge of 18 tons. It is necessary to shake the collecting bags after each charge; unless this is done they become clogged, causing back pressure, which slows the operation and injuriously affects both recovery and costs.

A report of one of the early operations at the National plant will give an idea of the work accomplished. I may state, however, that the work at the present day is much better.

Total Charge	Pounds
Skim	20,000
Coke	3,500
Soda	1,750

Assay Skim	Contents Pounds Zn	Pounds Pb + Sb
Zn, 16.50; Pb, 63.50; Sb, 6.00 per cent	3,300	13,700 + 1,200
Produced		14,990
Antimonial lead	.....	13,360
ZnO fume	3,825	
Assay		
Zn, 65; Pb, 14.65; Sb, 1.00 per cent	2,486	598
Residue (Soda Skim), 3,320 lb.		
Zn, 24.50; Pb + Sb, 9.8 per cent	813	325
	3,299	14,283

Some lead of course remained in the furnace.

While this process has not as yet been tried on a large scale with complex lead-zinc ores, I can see no reason why it should not prove successful in that field.

The additional cost of roasting would not be a serious matter. Most of the values would be taken up by the lead bath, and there would be only the residue or "soda skim" (amounting to from 10 to 15 per cent. of the original weight) to be sent to the blast furnace.

Experiments made in a small way gave returns that indicate a successful solution of the problem. The silica and sulphur in the roasted product do not seem to affect the zinc recovery. The silica would interfere only so far as it increased the amount of soda skim to be treated.

The Metallurgy of Zinc

Discussion of the papers of DORSEY A. LYON and SAMUEL S. ARENTZ, p. 789; S. E. BRETHERTON, p. 802; H. A. WENTWORTH, p. 809; and RICHARD D. DIVINE, p. 814.

GEORGE W. RITER, Salt Lake City, Utah.—We have every reason to believe that oil flotation will soon come into general use as a final guard against slime losses in concentrating mills, not only in the case of zinc ores, but also in the case of other semi-precious ores. It also promises much as a primary process for the concentration of minerals that are not adapted to gravity methods. Moreover, modifications of the process involving selective or preferential flotation are gradually becoming understood, and will make it possible to eliminate pyrite and other minerals occurring as undesirable impurities in zinc concentrates. And what perhaps is of equal importance, further modifications promise an effective method of separating zinc blende from ores that are chiefly valuable for copper, lead, and precious metals, thereby making an asset out of zinc which has heretofore been a liability.

The tendency of certain pulverized minerals to float on water, especially after having been in contact with grease, has been known to millmen for a long time, and was formerly something to be deplored and to be striven against. All at once, this tendency has become a saving grace; and now, like tardy converts to a new faith, we are zealous advocates of something we once despised.

I was once employed at a mine in the Tintic mining district of Utah—the old Eureka Hill property—where ores that were too low in value to warrant being sent to copper or lead smelters were treated in a combination mill. The ores contained gold, silver, lead, copper, zinc, manganese, arsenic, etc., in varying quantities and in varying mineral forms, mingled in a quartz and calcite gangue. The material was stamped fine and then concentrated on Frue vanners so as to make a smelting concentrate, after which the tailings were thickened and treated in amalgamating pans for the recovery of non-concentratable precious metals.

In the amalgamating pans, under the influence of heat and agitation, a dark greasy froth or scum sometimes formed on the surface of the ore mixture. This froth or scum was composed mainly of mineral sulphides, and its occurrence might have meant only casual losses in the previous concentration, except that the phenomenon was accompanied by faulty amalgamation of the precious metals and by excessive losses in the mill tailings. Grease and oil, present by accident, were the cause of the trouble.

Whether the intense vibration in the mill, due to the pounding of the heavy stamps, had caused small particles of grease to be shaken loose

from the heavy machinery, or whether the oil was merely lubricant that had passed through rock drills used in mining the ore, a single drop would spread over a large expanse of water and smear up a lot of pulverized ore. Having an experimental unit in our mill, we undertook to find out how to make the pulverized ore behave to our liking in spite of stray grease; and before we got through, we discovered a lot of facts concerning the behavior of greasy minerals under varying conditions as to heat, agitation, and weak chemical solutions. These facts ought to have pointed our way toward flotation as an economic process. They should have pointed further, toward the separation of flotative minerals from one another; because we observed that all minerals did not behave alike, and that sulphides of zinc were the most erratic of all. At that time, however, our results did not seem particularly significant, because zinc was then a liability rather than an asset, and what we were then trying to do was to prevent all forms of flotation; and our energies were directed toward finding lubricants that would be harmless and inactive in contact with ores, or which could be made harmless and inactive by means of reagents.

Zinc blende, when found in minute crystals mingled with pyrite and other sulphide minerals, eludes the simpler methods of mineral separation because it is not in itself unique as to density, crystallization, fracture, or behavior with chemicals. Its one distinctive difference seems to be in its behavior toward electricity. All of the other sulphides are good conductors; but zinc blende, except when contaminated with iron as an impurity, is a non-conductor. Upon this property rests the electrostatic concentration of zinc ores. A paragraph from Henry A. Wentworth's paper on Electrostatic Concentration or Separation of Ores[1] is worth repeating:

"There is an old experiment in physics where an electrified rod is brought close to a suspended pith ball. The pith ball is first attracted, clings for a moment to the rod, and is then vigorously repelled. As the rubber rod approaches the pith ball, a charge of opposite kind, so called, is induced on the side of the pith ball nearest to the charged rod, and as unlike charges of electricity attract one another and as the pith ball is very light, it moves to the rubber rod. But pith, though not a good conductor of electricity, does, because of the moisture contained, conduct electricity appreciably, and it soon becomes, as a whole, charged similarly to the rubber rod, and away it flies. This is the principle which is utilized in electrostatic separation, and to accomplish separation the differential property is the conductivity of the minerals involved."

In the same paper, Wentworth gives a list of more than 50 important conductive minerals; but zinc blende is placed in a list of non-conductors, along with quartz, feldspar, silicates, carbonates, sulphates, etc.

Whether or not it is anything more than a coincidence, experiment indicates that electrically non-conductive minerals are nearly neutral

[1] *Trans.*, xliii, 412 (1912).

toward oils and gases in the flotation process, and that minerals having the highest electric conductivity are most amenable to oil flotation. Mercury, galena, graphite, nickel ore, pyrrhotite, etc.—this list of conductive minerals,[2] in descending order, is the list of minerals most easily recovered as a flotation concentrate when in finely pulverized state. Using the coincidence as a working hypothesis and carrying the experiments further, we find that oil flotation is not restricted to sulphide minerals, but that tellurides, chlorides, oxides, native metals, and other native compounds, can be brought into the flotation class under the right conditions. Impure zinc blende, because of its iron impurity, also comes into this class. The flotation class is enlarged still further if useful carbonates be first converted into oxides by means of calcination. For example, zinc oxide derived in this way from smithsonite can be recovered by oil flotation if a proper oil is used. This bit of research is one that I have not yet carried to a final conclusion; but a mixture of linseed oil and turpentine is one combination that seems to be effective; and whether we say that we are floating the material as a froth or as a paint, the important point is to do the trick economically and to overcome interfering agents that occur in the worthless gangue.

It is my impression that before we finally get to the underlying principles of oil flotation, including the separation of flotative minerals from one another, the depths of electro-chemistry and electrostatics will have to be sounded pretty thoroughly. In commentating on the migration of particles suspended in liquids under the influence of electric currents, LeBlanc[3] reviews the work of Helmholtz and others, tending to show that the migration is due to the presence of an electric charge upon the portion of matter in question; that the direction of migration is influenced by the addition of small quantities of foreign matter, such as alkali or acid, in solution; that at the surfaces of contact of two dissimilar media—for instance, the contact surface of water and glass—an electrical charge or double layer must form; that the arrangement is entirely analogous to an ordinary electric machine, with only this difference, that whereas in one case a liquid rubs past a solid, in the electric machine a solid rubs past a solid; and finally Coehn's answer:

"If two substances are brought into contact with each other, the one possessing the higher dielectric constant will become positively charged."

Here again we have a striking parallel between the phenomena of electrolytic migration and the phenomena of oil flotation; and a question that arises is whether we are not justified in considering a flotation tank and its agitating device as a dielectric machine; the solution, with its

[2] Landolt-Börnstein-Meyerhoffer. Quoted in *Bulletin No. 548, U. S. Geological Survey*, Electric Activity in Ore Deposits, by Roger C. Wells, p. 25 (1914).

[3] LeBlanc: Text-book of Electro-Chemistry, translation by Whitney and Brown, pp. 157 to 160 (1907).

dissolved foreign matter, as an electrolyte; and the film of oil surrounding the mineral particle and any attached bubble of air or gas, as an insulator that enables the particles to retain their electrostatic charges, even in the presence of the liquid.

As between dry electrostatic separation on the one hand, and selective flotation on the other hand, the relative advantages may depend somewhat on the fineness to which the ore must be reduced in order to separate unlike minerals from one another. Oil flotation has the advantage of being adapted to mineral particles in a state of fineness approaching colloids, without calling for preliminary drying of the material, and without excessive installation and operating costs.

JAMES M. HYDE, San Francisco, Cal.—The flotation process being a new one in American ore-dressing practice, it may be well to outline briefly some of the factors that have to be taken into account in its application. The process depends upon certain physical phenomena quite different from those made use of in the ordinary water-concentration practice. Certain minerals having a metallic luster, particularly the sulphides, sulpharsenides, sulphantimonides, and tellurides, together with native sulphur and graphite, are easily wetted by oil, whereas the usual gangue minerals are not. There is an exception to this rule in that certain of the carbonates, particularly calcite and siderite, are rather easily oiled by the fatty oils, apparently because a certain degree of actual saponification of the oil by them takes place. Both the minerals which have been mentioned and oils also exhibit the phenomenon of readily attaching themselves to gases even in the presence of water.

The first successful application of flotation concentration on a large scale along anything like the lines now practiced was made in Australia by what was called the Potter process, in which the Broken Hill sulphide middlings, consisting of pyrite, blende, galena, garnet, calcite, and certain other gangue minerals, were introduced into a hot solution containing about 5 per cent. of sulphuric acid. The acid reacted upon the calcite, causing the evolution of carbon dioxide gas, to which the sulphide particles attached themselves and were thus buoyed to the surface and removed as a froth of concentrates. Following upon this work, installations were made in which a small amount of oil was added to the pulp, the amount of sulphuric acid was lessened, and the gases necessary for flotation were in part supplied by air beaten into the pulp by the violent agitation necessary to distribute evenly throughout the pulp the small amount of oil used and allow of a film coating of the sulphide particles with oil. This practice followed very closely along the lines disclosed earlier by Carrie J. Everson, and by Alcide Froment.

The process was first applied in this country on a large scale for the treatment of the ores of the Butte & Superior mine, at Butte, Mont. The original installation was made in a mill at Basin, Mont., where this

company was temporarily milling its ores. It was found both by this company and by another company working upon similar material from the same vein, that ordinary milling practice, making use of jigs, tables, and vanners, could not profitably make a higher recovery from these ores than from 50 to 60 per cent. of the zinc which they contained. Some further recovery might be possible by larger installations of vanners, but the product yielded by vanners was so low grade as to be salable at very small profit. The ore contains its principal value in zinc in the form of zinc blende, the average content of the ore ranging from 18 to 21 per cent. zinc. With this occurs a small amount of lead in the form of galena, and several ounces of silver—probably in the form of silver sulphide. The principal gangue minerals of the ore are quartz, rhodonite and rhodochrosite, and more or less altered inclusions of granite. The high content of zinc of this ore and the low recovery possible by water concentration alone made it necessary to find some new method for its treatment. Laboratory tests, accompanied by screen analyses and microscopic examinations, showed that the material was successfully treatable by flotation concentration, but that, because of the peculiar nature of the ore and the intimate association of the blende with the quartz, a special method of treatment would have to be worked out for it. The principal modifications that proved necessary were the tube milling of the tailings resulting from vanning and table operations; preliminary treatment of the slimy portion of the pulp with sulphuric acid and lime or some other coagulant; and a novel method of roughing and cleaning by flotation concentration. This ore presented a peculiar problem in that even the coarse crushing produced a high percentage of slimes which were particularly difficult to settle. Tests made upon them have shown that after a slimy pulp had remained in a quiescent condition in a Callow cone for 3 hr. the solids still in suspension contained as much as 14 per cent. zinc.

The practice that has been adopted by the company makes use of jigs and tables for the recovery of as much high-grade concentrate as is possible by the use of these machines and involves the treatment by flotation of all the slimes produced incidental to the crushing throughout the mill combined with reground tailings from the jigs and vanners.

The process which was worked out for the ore has been so efficiently administered by the staff of the company in their new mill, working at a capacity of 1,000 tons per day or more, that in the first six months of 1914, 193,000 tons of ore were treated with an average recovery of 89.3 per cent. of their zinc content, made into a concentrate containing 52.4 per cent. zinc. The elasticity of the process is shown by the fact that whereas in January this recovery was at the rate of 90.67 per cent. of zinc contained in the ore, the concentrate produced running 51.3 per cent. zinc; in June it had been altered so that the recovery went 88.7 per cent. zinc but the product had been raised to a grade of 54.6 per cent. zinc. There

is much greater profit in marketing high-grade zinc concentrate than low-grade material. On the basis that water concentration alone would profitably recover but 60 per cent. of the content of zinc in this ore, the figures given out by the company indicate that in this mill alone, during the first six months of this year, approximately 17,500,000 lb. of zinc were recovered which would have been lost, in large part at least, had it not been for the use of the flotation concentration process. A further advantage to the company has been gained by the ability to market its whole product in a higher-grade concentrate than would otherwise be possible.

S. A. IONIDES, Denver, Colo.—The dry chlorination treatment was started as a comprehensive scheme for saving zinc in these complex sulphide ores, and it proposes in one building to start with the ore and end with metals. The first step is drying, and crushing practically parallel to any other kind of drying and crushing, but with this distinction: it is not necessary to crush so fine. The only necessity is to expose one surface of each metallic particle, and not necessarily to isolate it, and the chlorine which comes in the second stage will attack that and be able to pierce it. Working at low temperature the sulphur gets no chance to coagulate, but the reaction between chlorine and the metallic sulphides, resulting in the formation of metallic chlorides, cannot be carried to a conclusion, and so a second stage is necessary, which consists of a light chloridizing roast, between 200° and 400° C. In the first step I should have mentioned that silver and probably gold are chloridized in addition to the base metals, copper, lead, zinc, and iron. The ferric chloride is not wanted in particular, and in ordinary practice it is found better to decompose this by the light roast I have mentioned, and the chlorine, freed from the ferric chloride, attacks the sulphides from the other metals and converts them into chlorides. The roasting ends the dry part.

After roasting, the ore is leached with hot water, and this will dissolve all of the metals with the possible exception of the gold. If there is any ferrous chloride or other reducing agent present the gold will not be dissolved. It will be carried out. Silver chloride is readily soluble in solutions of other chlorides, lead chloride in hot water, and zinc chloride and cupric chloride in water of any temperature. After thorough agitation the gangue is filter-pressed off, and the solution containing the metallic chlorides then goes to the refinery. The refinery follows the ordinary course of wet refining. Metals are precipitated on one another, the gold and silver on copper, the copper on iron, the lead on zinc, and the iron is removed with zinc oxide. This leaves a solution containing pure zinc chloride, which is then evaporated to dryness, fused, and electrolyzed in a fused condition, giving zinc and chlorine ready for re-use. The one point I would make about the process is that the loss should be very low. There is just one point in the process where loss can occur in-

stead of three, four, or more, as in the ordinary retort and other processes. The one point is in the actual chloridizing. If that is not complete some zinc will go out with the gangue, but with the rest the only chance of loss is mechanical, and we hope to be able to keep that low.

S. S. ARENTZ, Salt Lake City, Utah.—During the past six months I have met a number of men more or less interested in the Rankin-Westling process for the wet treatment of all classes of sulphide ore. These men appeared to be so enthusiastic about this process that a word here will not be out of the way. The process consists in treating the ore with nitric acid in a closed retort. Steam is applied under pressure. The ore must contain about 7 per cent. sulphide. The gases and vapors driven off are caught in various receptacles. The gases caught include NO and NO_2 and contain nearly or all the nitrogen present in the nitric acid used in the treatment. The solution will contain, as sulphates, all the metals present in the ore. The treatment of zinc is particularly interesting to us to-night, and I have copied verbatim an extract from the Rankin pamphlet.

"After silver, bismuth, arsenic, iron, aluminium, copper, cobalt, nickel and cadmium have been removed, one can

"(a) if metal is desired, add to the cold sulphate a little less than theoretical amount of CA (OH_2) necessary to precipitate all zinc decant, filter and wash and treat the mixed hydroxide of ZN and CA SO_4 precipitate, electrolytically with an E. M. F. of less than three volts in a Rankin-Westling cell, or

"(b) if the oxide is desired treat the ZN (OH_2) and CA SO_4 ppt. with SO_2 in a closed tank to get ZN ($H\ SO_3)_2$, decant, filter and treat the acid zinc sulphite solution as given below for last part of our improved method, *i. e.*,

"(c) to the sulphate solution in a closed tank add the proper amount of CA ($H\ SO_3)_2$ plus some free SO_2, to obtain a precipitate of CA SO_4 and a solution of sulphates and ZN ($H\ SO_3)_2$ in dilute H_2SO_3; decant, filter, wash under pressure, into a closed precipitating vessel. Heat and from this solution release the first molecule of SO_2, thus obtaining a solution of remaining sulphates and a precipitate of zinc sulphite. Wash with H_2O; decant and filter; press the ZN SO_3. Remove zinc sulphite to a Rankin-Westling rotary tant-iron retort and drive off SO_2, and obtain technically pure zinc oxide. While still hot, drop into cold distilled water to render less crystalline, if desired.

OLIVER C. RALSTON,* Salt Lake City, Utah.—I am asked to take up in general the hydrometallurgy of zinc. I will try to state the situation rather briefly. At the present time at least 17 different corporations or individuals are conducting experiments larger perhaps than the test tube—that is, tests of 100 lb. or more—and doubtless a great many of the large zinc corporations are quietly carrying on work, although it is very hard to find out. Of those 17, over half are processes consisting in general of leaching with sulphuric acid, with electrolytic precipitation in view, and a few are leaching with hydrochloric acid, or metal chlorides, with

*Non-member.

electrolytic precipitation in view. The other processes, two of which you have heard to-night, the Rankin process, just discussed by Mr. Arentz, and the Bretherton process, as well as the dry chloridizing processes, are somewhat different. There seems to be no difficulty in getting zinc into solution. There is one plant in Russia operating commercially, I think, on about 30 tons of ore per day, consisting of a limestone having 6 per cent. of zinc as carbonate, and they actually leach that out with sulphuric acid and make money on it. They have a special method of precipitation with a depolarizer of iron. In England is a notable plant, the Bruner-Mond Alkali Works, which has been going for years, using up a calcium chloride waste as a source of chloride ions, and getting the zinc into solution as a chloride from certain roasted zinc ores. They have been precipitating electrolytic zinc supposedly 99.6 to 99.96 per cent. pure, and I was informed to-day that they get 1½c. to 2½c. per pound extra for that pure zinc. In Germany for years there has been one plant, designed by Hoepfner, precipitating zinc electrolytically. The exact details of the industrial electrolysis of zinc sulphate, or of zinc chloride solutions, have never been published. They are of such a nature necessarily that if they were to be published these companies would have too much competition, and destroy their own trade—supposedly. Very recently work has been going on in California at Bully Hill, in which very good cathode deposits of zinc have been made, but we understand that to precipitate electrolytically requires a great deal of capital, and it is practically a rich man's process; it does not apply to any small work in zinc. That I think covers pretty well the present situation in regard to zinc hydrometallurgy. Recent work in England has shown that zinc can be precipitated in the presence of iron by an application of certain retardation and over-voltage phenomena, getting a very good deposit of zinc, and, moreover, the over-voltage prevents many bubbles of hydrogen, which formerly caused trouble in the way of trees, spongy zinc, and other such deposits of the zinc, which are undesirable.

D. A. Lyon, Salt Lake City, Utah.—The idea of igneous concentration is illustrated by the Fink process, with which some of you who are present are familiar. In this process, the furnace is a revolving cylinder which somewhat resembles a trough copper converter. After it is charged, the furnace is revolved while being heated, and air is blown through the charge. In this way, the zinc is oxidized. As before stated, this process illustrates what is meant by igneous concentration. This manner of recovering zinc from its ore appeals to a great many. They believe it is the ultimate solution of the zinc problem. However, one of the disadvantages of igneous concentration is that after you get your precipitate, or whatever form you get your zinc in, it is liable to be very fluffy and hard to handle.

Mr. Swart.—This morning I was introduced to a gentleman in the lobby, and he looked at me and said: "Oh yes, you are the zinc man," in

the same sort of a tone as though I might be the ice man or the milk man. I suppose that means that they feel sorry for the man who is in the zinc business. We feel sorry for ourselves, and we have good reason to do so. The fact is that no one can predict just what is going to happen in the metallurgy of zinc. The losses discussed in the paper just read are not all of one kind and cannot all be corrected in the same way. Some of them are hopeless losses, I am afraid. The loss of zinc in smelter slag is one decidedly important loss, but in that same slag iron is lost as well as zinc. Is it fundamentally any worse to lose the zinc than the iron? This iron may not be worth as much per pound, but there is more of it and it is probably just as hopelessly lost. We ignore this loss because our iron supplies so enormously exceed our zinc supplies that the loss of the one seems trivial compared with the other, yet it is all a matter of degree, not of kind. We have had to make a slag from iron to get our lead metal and we have had to use zincky ores for the same reason.

There are some new things in the metallurgy of zinc, and I was glad Mr. Lyon mentioned them. I feel pretty well satisfied, for example, that igneous concentration is going to work well, but it is not going to be universally applicable. There doesn't seem to be anything universally applicable to zinc ores. Igneous concentration is being tried out now, and in my opinion a decided step in advance is being thus taken.

LAWRENCE ADDICKS, Chrome, N. J.—I noticed one of the speakers in the discussion said that the very pure zinc commanded a premium of 2½c. a pound. I had always understood that the difficulties in the way of producing electrolytic zinc were that the cathode was so rough that it could not be marketed without remelting, entailing a metal loss almost as high as that of the old fire process, and that you couldn't get any more for the product than you could get for ordinary spelter. The price of zinc is so low that it is very difficult to justify the cost of an elaborate process. I would be interested in knowing for what purpose the zinc which commanded so high a premium was used.

MR. SWART.—There is a limited market for high-grade zinc at an excess price, but it is a very limited market. In case any considerable amount of spelter were produced and thrown on the market by such processes as have been described, the premium would surely disappear. The premium is not 2½c. unless in exceptional cases in retail or special lots, but is considerably less than that on larger contracts, and even these contracts do not constitute what may be called the metal trade. There is also another thing of interest, and that is, that spelter made by electrolytic processes, while it may be chemically pure, or very nearly chemically pure, may still not have all of the necessary physical qualities. If you go into the market to-day to buy spelter, for instance, for making a special grade of spinning brass, and are willing to pay a premium for it, you can get it, but it won't be an electrolytic spelter. It will be made by the New Jersey Zinc Co. from ore found in Virginia or Tennessee, and

electrolytic spelter, of which quite a little has been made and tried in Europe, does not apparently give the same results, in spite of its greater purity. This may be only one of those peculiar trade prejudices based on custom and inertia, but it exists and will be difficult to overcome.

S. E. BRETHERTON, San Francisco, Cal.—In regard to the special price of zinc made by the electrolytic process, I was informed by the chief chemist at Syracuse, N. Y., two years ago, that parties were making zinc in England by electrolytic precipitation, for which they were then getting from 1½c. to 2c a pound premium, which I think verifies, or helps to verify, the statement made by Mr. Ralston, and I took his statement for granted because such a man could not afford to make a misstatement. At that time I was looking into the recovery of ammonia, and the losses of ammonia in the Solvay process, where, as you know, they use carbonates of ammonia for making carbonate of soda and chloride of lime. The last product is mostly thrown away. I wish to state further that to-night I simply described the ammonia-carbon dioxide process very briefly, but it is fully described in the *Transactions*.[4] I thought I would mention this in case any one wished a detailed description of it. In that particular article I laid stress on the fact that the process I described briefly to-night is not suitable for ores containing silicate of zinc, or silicate of copper, or ores containing an appreciable amount of arsenic.

G. B. WILSON, Salt Lake City, Utah.—I would like to ask what this particularly high grade of zinc is used for. I presume it is for the manufacture of the salts of zinc.

MR. BRETHERTON.—That I cannot say. I simply know that the gentleman made that statement.

W. McA. JOHNSON, Hartford, Conn. (communication to the Secretary*)—In the production of a metal there is a certain loss. The losses of zinc, as scheduled so clearly by Mr. Lyon and Mr. Arentz, are generally higher than the losses of other metals, such as gold, silver, copper, and lead, for two interdependent reasons: zinc has sold for a 30-year average of but 5c. per pound, and is plentiful geologically as compared with the other metals.

The peculiar nature of its reduction necessitates a commercial minimum of 22 per cent. for such carbonates and of 30 per cent. for such sulphides as go to the retort plant, since the cost of coal and labor per ton of ore retorted is high. Accordingly, concentration processes—wasteful in metallic values, since blende is hard to concentrate, but cheap in operation and in capital cost—must be used to raise the grade of the ore so that

[4] *Trans.*, xlvii, 82 (1913).

*Received Aug. 20, 1914.

it will be rich enough to stand the $20 to $30 aggregate cost of freight, metallurgical losses, and treatment charge per ton of ore. It is a rigid principle that it pays to waste mineral to attain the maximum commercial usufruct.

In short, we see that if zinc were not so abundantly distributed geologically that it could stand heavy losses in mining, milling, and retorting, and other commercial and adverse factors, it would cease to exist as a metal of prime commercial importance; for any increase in price cuts off consumption more markedly in the case of zinc than in the case of other metals, since, generally speaking, its use is not indispensable.

At any selling price, however, below 6c. per pound, society can afford to use spelter for many purposes, chief of which is galvanizing. There are two main reasons for the enormous growth of consumption of spelter in galvanizing, for which 60 per cent. of the spelter is used: People are building things in more permanent fashion each year and are therefore using galvanized iron and steel. Parenthetically, let me state my belief that some day steel bridges and structural steel in a large way will be galvanized. Also our use of sheet zinc for roofing material will be comparable to foreign consumption. Furthermore, the increase in the selling price of lumber has enormously increased the use of galvanized sheets. Considering conditions from several angles, we can expect zinc to increase in consumption in the future as it has done in the past provided the price does not exceed 6c. per pound. Confirmatory of this *a priori* reasoning, we find that if we extrapolate Mr. Siebenthal's curve of compensated spelter consumption,[5] the consumption of spelter in the United States by 1924 should equal about 850,000 tons. It would be impossible to produce all this amount of metal in the retort furnaces as now operated, and besides the ore for this stupendous tonnage can come only by improved mining and milling and metallurgical practice.

Let us consider now the influence of the Johnson process in helping to furnish this large tonnage. I had best introduce this side of the question by saying that with power conditions and coal conditions as they are in North America, the Johnson process will not try to compete with the retort process on ores high in zinc and low in other values. The reason is plain: the cost of retorting a ton of Joplin ore is $10, and the cost of retorting a ton of Leadville or other Western sulphide ore is $12.50. In electric smelting the reverse is true and the cost of treatment is less as zinc tenor decreases. That is, the cost of treatment increases with increasing tenor of zinc. Let us assume as exactly true the probable truth that 100 lb. of slag-making materials can be smelted for 5 kw-hr., lead for 5 and matte for 10 kw-hr., and that 100 lb. of zinc can be smelted electrically for 125 kw-hr. Then two typical zinc ores of extreme type would show up as follows:

[5] *General Report Zinc and Cadmium in* 1912, U. S. Geological Survey.

Low Zinc-Lead Ore, Roasted

Zinc, 20; lead, 15; matte, 10; slag, 35 per cent.

Zinc	400 lb. @ 125	500
Lead	300 lb. @ 5	15
Matte	200 lb. @ 10	20
Slag	700 lb. @ 5	35
Kw-hr. per ton of ore		570

High-Grade Zinc Ore, Roasted

Zinc, 70; lead, 2; matte, 1; slag, 7 per cent.

Zinc	1,400 lb. @ 125	1,750
Lead	40 lb. @ 5	2
Matte	20 lb. @ 10	2
Slag	140 lb. @ 5	7
Kw-hr. per ton of ore		1,761

To simplify calculation I have assumed 20 per cent. of oxygen in each case.

If we pay 3 mills per kilowatt-hour, energy cost will be in the first case $1.70 and in the second $5.28; at 5 mills per kilowatt-hour, in the first case $2.85 and in the second $8.80. The prohibitive charge of $8.80 for electrical energy in the case of Joplin ore can be seen at a glance.

I was fairly well convinced of the correctness of my own reasoning as exemplified above, but when it was corroborated by C. A. H. deSaulles, of the American Smelting & Refining Co., who has extensive practical knowledge of the zinc business, I felt absolutely sure that the proper line of development of the Johnson electric zinc furnace lay in zinc-lead smelting. If we add to the above the fact that there are plenty of zinc fluxing ores containing lead, copper, gold, and silver, with lime and occasionally fluorspar, and consider that our lead recovery will be higher than that of the lead furnace, we see the wisdom of starting zinc-lead smelting rather than electric zinc smelting pure and simple. It is indubitable logic. The fact that there are possibilities of saving labor in electric smelting has also bearing.

Exactly how all this electric zinc-lead smelting will affect the concentrating of ores is unknown. But it can be prophesied when crude ore is not used that the procedure will be to make roughly a zinc-lead middlings with considerable included galena, subjecting the tailings to the flotation process after grinding, then make other sulphide concentrates, giving clean tailings. Any part of the milling process that would naturally give a product high in zinc would be used, and this would go to the retort plant, whereas any stuff high in total zinc plus lead would go to the electric furnace plant. The marvelous perfection that the flotation proc-

ess has reached at the Butte & Superior mill as a "clean-all" on tailings inspires belief in its large future sphere of usefulness. The electrostatic separator has several avenues of further success, especially when the copper sulphate treatment to make zinc sulphide superficially conductive is employed. Igneous concentration, as talked about by Mr. Clerc and done by the late Mr. Truax, has certain possibilities, but I do not look for it to come about at once nor to be of much importance.

Hydrometallurgical processes will be used some day when zinc ore is scarce and chemical knowledge is so far advanced that laborers can make a "Group II A and Group II B separation" respectively, but that day is far distant, for quantitative analysis writ large does not spell metallurgy.

The improvement in the retort practice will surely become manifest, as can be judged by the increased tonnage that the plants in Kansas, Oklahoma, Illinois, and elsewhere have attained in the past 10 years. A charge of 16,000 lb. of roasted Joplin ore to a 320-retort furnace would have been unthinkable in 1903, when 13,000 lb. was the standard charge, yet to-day 16,000 lb. is normal. Unquestionably machine charging and discharging is about to be practicalized. I have always been inclined to larger retorts, but other practical retort men differ with me in this respect.

Along with this improvement in retorting and the advance in electric smelting and inmillingmethods, we can expect a combination of processes, each doing its own duty at a maximum efficiency. Considering the probable increase in demand in the United States and the fact that the European output, amounting to 600,000 short tons, is produced in centers now distressed by war, we can foresee that an enormous production will be in demand in the United States. This can only be met by the united effort of all zinc producers on this continent. The paper under discussion will contribute to the carrying out of such a task, and as such it is worthy of respectful attention and praise.

Biographical Notice of Louis Janin

BY R. W. RAYMOND, NEW YORK, N. Y.

(Salt Lake Meeting, August, 1914)

THE death of Louis Janin, which took place Mar. 6, 1914, at Santa Barbara, terminated the earthly career of almost the last survivor, and

LOUIS JANIN

one of the most illustrious members, of that group of American mining engineers and metallurgists who, in the last quarter of the nineteenth century, directed the development of the mineral resources of the Pacific slope.

Louis Janin was born in 1836 at New Orleans. His grandfather, an officer in the French army, had emigrated to this country, with his family, at or before the outbreak of the French revolution. His father became an

eminent member of the New Orleans bar. He was the eldest of six sons, two of whom (Henry and Alexis) became, like him, mining engineers and metallurgists.[1]

Of the other three, one, as I am informed, entered the Confederate army during the War for the Union, and died in battle; another, following his father's profession, achieved distinction as a lawyer in Washington, D. C. The third, Albert, the only survivor, is said to be now the manager of the Mammoth Cave property in Kentucky.

As I have explained more fully in my Biographical Notice of his brother, Henry Janin, their father was professionally engaged in the litigation (preceding the enactment of any U. S. laws governing mining titles on the Pacific slope) over the New Almaden quicksilver mines of California, and had thus become acquainted, on better authority than that of general rumor, with the immense immediate and prospective mineral wealth of the American West. No doubt he foresaw the opportunities of profit and honor which the development of this wealth would offer to trained experts; and this explains, in my judgment, the sudden change in his plans for the eldest two of his sons, Louis and Henry (only two years apart in age), who, in 1856, after completing their Sophomore year at Yale University, sailed for Europe, and entered in October, 1857, the Mining Academy of Freiberg, Saxony, where they studied for three years. That was, perhaps, the period of Freiberg's greatest glory—not because the Academy had not had before, or has not had since, good reason to boast of great leaders and teachers; but because the fame of that group of men, as scientific investigators, technical authors, skillful and thorough instructors, and sympathetic, influential critics and guides of practice, was world-wide and well-nigh unrivaled. The mere enumeration of the names of Weisbach, Breithaupt, Cotta, Gaetschmann, Scheerer and Fritsche—not to specify others of that splendid corps, who worthily took their places afterward—is enough to explain the pre-eminence which Freiberg then enjoyed.

The Janins utilized thoroughly the benefits of such instruction, becoming in particular intimate associates of the genial and brilliant Cotta, with whom, after finishing their course at Freiberg, they took a long professional trip through Bohemia and Hungary, of which Cotta subsequently published a highly interesting account. It was when they returned from this journey for a farewell visit at Freiberg, in October, 1860, that I formed with them the acquaintance which became a life-long friendship.

After a brief supplementary course at the Paris École des Mines, the two brothers sailed for America in 1861. As I have said already, one of their younger brothers entered the Confederate army. But, so far as I can now recall, neither of them passionately took part in the conflict

[1] See my obituary notice of Henry Janin, *Bulletin* No. 53, May, 1911, p. xxviii.

of opinion which preceded the conflict of arms. No doubt they sympathized more or less with the South; but they had been abroad during the years of exciting controversy which had developed State pride in the South from a traditional sentiment into a fanatical, belligerent religion. Moreover, they were not particularly interested in the State of Mississippi, or the cultivation of cotton and sugar, or the institution of slavery, intimately connected therewith. Their thoughts had long been turned to the new, wide, free region further West, for the service of which they had been preparing themselves by years of study and travel. In fact, they were already enlisted for that war which went on *pari passu* with the War for the Union, and was waged by an army of prospectors and miners, for the physical conquest of a new Empire, while other hosts were battling to determine its political destiny. It is not surprising, therefore, that the two brothers soon found their way to the Pacific slope, and into active practice as mining engineers. One of Louis Janin's earliest experiences was an encounter with Apache indians, who attacked in Arizona the exploring party of Col. Butterworth, of which Janin was a member. The story is told by J. Ross Browne in *Harpers' Magazine*, vol. xxx (1864–65), p. 287 *ff.*, and in a book by the same author, entitled *Adventures in the Apache Country*. It shows young Janin to have behaved with cool and intelligent courage.

After a brief engagement as Superintendent of the Enriquita quicksilver mine in the Coast Range of California—a position in which his brother Henry was first his assistant and afterward his successor—Louis seems to have turned his attention to the mining and metallurgy of silver ores, for which his studies abroad had specially prepared him. The pressing problems which called at that time for the aid of trained experts were centered around the Comstock lode, the rich bonanzas of which were extracted with difficulty and reduced at enormous loss. Such superintendents as Deidesheimer invented methods of mining suited to local conditions; but the treatment of the peculiar auriferous silver ores of the Comstock mines required long and patient experiment, the history of which has been well told by A. D. Hodges, Jr., in his paper, "Amalgamation at the Comstock Lode, Nevada," etc. (*Trans.*, xix, 1890, p. 195), and in an article in the *Mining and Scientific Press* of San Francisco, May 21, 1910.

I shall do no more than briefly indicate the nature and value of the services of the Janins, and particularly of Louis Janin, to the metallurgy of the Comstock and other Pacific districts, not only by their laborious investigations and ingenious inventions, but also by their unremitting attention to details and discipline in practice, which diminished the leakages and losses, even of imperfect processes.

Already in 1863, according to Mr. Hodges, Louis Janin was metallurgist of the "Mexican" mill, at Empire City, Nev., where the auriferous

silver sulphides of the Mexican mine on the Comstock lode were treated. Some three years before, the Washoe process of pan-amalgamation had been introduced by Almarin B. Paul, a mill manager of genius and experience, who combined in this method the principles of the Mexican patio process with the use of pans, such as he had used in the gold mills of California. Operated with regard chiefly to mechanical capacity for speed, endurance, and large production, the Washoe process showed magnificent mills and machinery, and yielded large dividends of profit from the simple sulphides of the richest ores. But a large proportion, even of this material, was lost as "slimes," being ground to crystalline powder, the form and fineness of which caused it to float away, escaping the pan-amalgamation. To some extent, this loss of slimes was due to a chemical cause, namely, the presence, even in the rich ores, of multiple sulphides, which were not easily and quickly decomposed under the rude and rapid pan-grinding so as to yield amalgamable metal. An additional, and perhaps, in the aggregate, a larger loss was incurred in the sands or tailings from leaner ores, the valuable portion of which, enveloped in or adherent to particles of siliceous gangue, did not float, like the slimes, but, escaping the pan-amalgamation, whether for mechanical or for chemical reasons, flowed away into the tail-race. And finally, the tail-race received also many particles of amalgam already formed, but not aggregated so as to be successfully collected in the ordinary settling apparatus. These various sources of loss were not unknown, or totally disregarded; but the abundance of profits and the pressure for dividends required the exhaustive efforts of mine and mill managers, and little was done to determine or remedy current leakages, beyond the impounding of tailings and (so far as possible) of slimes, in yards and reservoirs. One further cause of this neglect was the lack of complete frankness and co-operation between the mines and the mills. Even when the same company owned both mine and mill, the latter might receive also "custom" ore for treatment; and it was not regarded as good policy to let customers know exactly what the mill extracted. The delivery of a certain percentage of the sample-assay value (and a pretty low percentage, too, it used to be!) settled the transaction. Of course, when the mills afterward became the property of a separate set of owners, the betrayal of their economic efficiency was still less desirable. And what is not to be told to certain other persons had better not be known to anybody. At all events, Mr. Hodges says that in his active experience in Nevada, he knew of only two silver mills in that State, treating silver ores without roasting, which checked their own work so as to know exactly what they were doing.

This was the situation in 1863, when Janin was at the Mexican mill, where (upon his initiative, as I naturally presume) the "Freiberg" system of amalgamation in revolving barrels, instead of pans, was tried, first

upon raw tailings, with only partial success, and afterward upon dried and roasted tailings, with better results. The latter practice became known afterward as the Freiberg process, and was widely practiced, for a while, on the Pacific slope in the treatment of "refractory" ores (*i.e.*, multiple sulphides, antimonides, arsenides, etc.); the trouble of drying the pulp being avoided by dry crushing in the mill. But the expense of roasting, added to the extra cost of dry crushing, proved fatal in the end to the Freiberg process; and, though it died hard, it died. Janin realized its economical weakness; and when, in 1864, the technical ability he had shown at the Mexican mill led to his appointment as Superintendent of the Gould & Curry Co., he gave his attention at once to the devising of a practicable humid process which would secure the chemical effects of roasting without involving that fatally expensive feature.

With what indomitable courage, patience and skill he prosecuted this inquiry, first for the Gould & Curry Co., and afterward, with greater freedom and daring, in a mill of his own, with no Board of Directors to prohibit experiments and demand dividends; how he attained technical success, only to see his commercial reward swept away by a flood which carried off the richer half of the 100,000 tons of tailings and slimes in which he had invested his capital; how he bravely overcame this disaster, and established, with his brothers, Henry and Alexis, a large and profitable business in the treatment of tailings and slimes by a modified Washoe process, without roasting, and by the intelligent use of well-known chemical reagents in the pan; how this process, not being patentable (since mere genius and industry and patience, however unusual, are not legally new), was copied by others, at least so far as to make mining companies refuse to sell their tailings and slimes, so that the Janins were at last driven out of business—all this is told in detail by Mr. Hodges, in the Institute paper already cited. And I might add to his story an additional chapter, telling how the establishment of railroad connections and the establishment of great smelting plants, where varied materials from many regions could be advantageously collected and mixed for economical reduction, has revolutionized many metallurgical situations of 50 years ago, and, in many cases, solved by simply erasing the problems over which the pioneers of American metallurgy spent their lives and broke their hearts.[2] But no such subsequent obliteration of immediate results should be permitted to obscure our grateful recognition of the work of those men who were the leaders of their generation.

I believe that, after leaving Nevada, Louis Janin was for some years manager of mines in Mexico; and I know that, somewhere in the '70's, he was employed for a year by the Japanese government, to make plans concerning the operation of certain gold and silver mines in Japan.

[2] See, in this connection, my Biographical Notice of Henry Janin, *Bulletin* No. 53, May, 1911, p. xxviii.

That was the period of Japanese progress when the two parties were so evenly balanced that the party of progress was strong enough to inaugurate new enterprises, and its opponents were strong enough to hinder their prosecution. I have often thought that this circumstance explains the erroneous notion, entertained in those days by many of us, that the Japanese were shallow and fickle. I remember Janin's humorous account of his experience among them—with what overwhelming courtesy, amounting to reverence, he was treated; how promptly his handsome salary was paid in gold; how freely all facilities for his work were provided; with what effusive gratitude and compliment his report and plans were received—and how absolutely nothing happened afterward, until, ashamed of drawing pay while doing nothing, he resigned and came home. Other American engineers had, in those days, a similar experience.

After his return from Japan, Mr. Janin enjoyed for many years an extensive practice as consulting engineer and expert, especially in connection with mining lawsuits involving questions of economic geology. Being not only a practitioner of long and varied training, but also a constant student and critic of technical literature, and characterized by imperturbable good nature and self-control, he was one of those expert witnesses whom wise cross-examiners handle with courtesy and caution. As an adviser, his clients found him upright, loyal and vigilant. His versatility, wit and comprehensive culture made him a fascinating companion in the old Union (later the Pacific-Union) Club of San Francisco, which was for many years his business address; and his generous helpfulness to younger mining engineers made a host of them his grateful debtors.

A little more than 20 years ago, Mr. Janin established his residence on a ranch at Gaviota, near the Mission of Santa Ynez, in southern California, where he lived with his family, except when temporarily called away for professional work. I never visited him there; but I had been repeatedly a guest in his Nevada home; and I could appreciate the desolation which fell upon him with the death of his wife, some years ago. Other afflictions he was called, in later days, to bear—including partial blindness and physical disability of other kinds, which must have made life a burden to him. Yet, if I may judge from his occasional letters to me, he carried that burden bravely and patiently to the last, crowning an honorable and useful life with a serene old age and a victorious death.

Louis Janin joined the Institute in 1872. He died at Santa Barbara, Cal., Mar. 6, 1914, of heart disease, after an illness of about ten days.

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